CYANIDE PRACTICE IN MEXICO

Earle B. Patten,
Philipsburg,
5-26-16. Montana

CYANIDE PRACTICE

IN

MEXICO

BY

FERDINAND McCANN

PUBLISHED BY THE
MINING AND SCIENTIFIC PRESS, SAN FRANCISCO
AND
THE MINING MAGAZINE, LONDON
1912
(Printed in U. S. A.)

PREFACE

The following pages consist largely of extracts from a book written and published by the author in Spanish, entitled 'Beneficio de Metales de Plata y Oro por Cianuracion', to which has been added new material in order to bring the practice in various Mexican mills up to date.

The general appreciation of the Spanish volume, and the fact that the Mexican practice deals principally with the cyanidation of silver ores, has caused the author to believe that the present volume, describing in detail the practice in Mexican mills, will be an acceptable addition to the literature on the subject.

I wish to acknowledge my indebtedness to the various persons mentioned in the text as having furnished me data and information, and especially to M. H. Kuryla, D. L. H. Forbes, and Bernard MacDonald for having written articles for publication in this volume.

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American Club, Mexico, D. F., Aug. 14, 1912.

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cost of the operation would be prohibitive, requiring either to increase the number of tanks for the treatment of the pulp, or to diminish the quantity treated. But by classifying the sand and slime, and treating them separately, each may be treated by a different system which yields the maximum percentage of extraction at the minimum cost, and combined give very satisfactory results.

The recent improvements in the apparatus for filtering slime are so great that the majority of the modern mills are installing regrinding machinery, in order to convert all of the sand into slime, and treat the whole product of the mill as such. The companies which follow this treatment claim that the increase in cost of treatment, due to regrinding the sand, is compensated, by the greater extraction obtained, by the shortened time of treatment, and by the lowering of cost of treatment due to the fact that where only one method of treatment is required, the cost of installation and maintenance is less. On the contrary, in dealing with concentrate and tailing already formed, they claim that it is more economical to treat them without further grinding; as the installation and operation of regrinding machinery would be more expensive than to treat the tailing for a longer time or even with stronger solutions. However, this is a matter easily determined in each particular case.

Various Preparatory Treatments Before Cyanidation.

Many ores contain substances that require a preliminary treatment before cyanidation in order to allow a greater percentage of extraction and a lower cost. The ordinary preliminary operations are the following:

Roasting.

Ores containing clay are so impermeable that it is impossible to filter them in percolation tanks without previous roasting. This roasting, by driving off the water of hydration, converts the clay into brick, which may be ground and afterwards treated in percolation tanks without difficulty. In roasting clays containing lime and magnesia the temperature should not exceed 300° F., as otherwise they may be converted into cement which, after being ground and mixed with water, may harden in the tanks.

Ores containing tellurides yield such low extractions when treated raw that it has been considered necessary to give them a preliminary roast, with the result that in Kalgoorlie, where this treatment is practiced, the extraction has increased from 60% on raw ore to 93% treating the roasted product. The roasting may introduce soluble metallic salts which would decompose potassium cyanide and thus increase the cost of treatment, consequently it may be necessary to give the roasted ore a water wash in order to remove these salts before cyanidation. Several years ago, in Cripple Creek, Colorado, mills were built to treat the tellurides of that

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mills in El Oro, State of Mexico, which formerly had amalgamating plates, have removed them, as they found that the free gold particles in the ore were flattened in the tube-mill into sheets so thin that they dissolved immediately in the cyanide solution. Amalgamation may be applied to the ore either before or after cyanidation, each method having certain advantages and disadvantages.

Amalgamation before cyanidation.—The advantages of this method, whether battery or plate amalgamation, consist in the fact that the pulp, without extra expense, is made to flow evenly, and in small quantities, over the plates, thus presenting the best conditions of contact between the gold or silver and mercury. The disadvantages are that small globules of mercury or amalgam, loosened by the jarring of the stamps, pass into the cyanide treatment tanks with the pulp, and, as they are insoluble in the cyanide solution, are discharged with the tailing, thus causing a loss not only of the gold and silver originally contained in the amalgam, but also of a certain quantity in excess, which the amalgam precipitates from the cyanide solution in the treatment tank. Furthermore, should the pulp contain any coarse grains of free gold or silver coated with a film of sulphide which prevented their amalgamation before cyanidation, such coarse grains would enter into the treatment tanks, and on account of their size, would not be dissolved in the time during which the pulp remained in the tanks.

Amalgamation after cyanidation.—This method is more often used in connection with dry than with wet grinding. Its advantages are that all particles of gold and silver which still exist in the metallic state will have been thoroughly cleansed by the action of the cyanide solution, and present the best conditions for amalgamation. Furthermore, as no amalgam is introduced into the cyanide solution the possible losses by precipitation, mentioned in the preceding paragraph, are eliminated.

The disadvantage of this method consists in the impossibility of securing the slow and regular discharge of the tailing over the plates, which is necessary for perfect amalgamation, without special storage tanks and discharging apparatus, as it is evident that if the treatment tanks were discharged directly on the plates the pulp would pass so rapidly and in such great quantity that almost nothing could be saved. The installation and operation of such a plant would greatly increase the cost of treatment. Another disadvantage of battery or plate amalgamation, whether practiced before or after cyanidation, is the possible theft of amalgam, which causes great loss. Loss by theft is more easily guarded against when the precipitation of all valuable products is conducted within a locked room.

Pan or Patio Amalgamation.

At times such treatment gives good results; as in the case of an ore which does not yield a satisfactory extraction by cyanidation,

it may be possible to extract the greater portion of the gold and silver by amalgamation, either with or without a preliminary chloridizing roast, and afterwards treat the amalgamation tailing by cyanidation with good results. In such a case it will be necessary to give the amalgamation tailing a water wash, to remove the cyanicides, before cyanidation.

Systems Based on the Method of Applying the Cyanide Solution.

There are various systems of cyanidation in use, based on the method of applying the cyanide solution, and the proper system to select for any particular ore will depend on the character and value of the ore, its composition and reactions, the situation of the property (to determine the relative cost of installation), the class of laborers and the wages paid them (to determine the advisability of installing labor saving devices), the kind of motive power and its cost, cost of fuel, analysis of water supply, etc. The ordinary systems of treatment are as follows:

(1) **Treatment by percolation.**—In this system the sand is placed in tanks where it receives consecutive washes of water and cyanide solution, each wash being allowed to percolate, or drain, through the sand until it is found, by assaying samples of the sand taken at regular periods, that a satisfactory extraction has been obtained. The tailing is then discharged from the tank, by shovels, by machinery, or by means of a stream of water, and the tank is then ready for a new charge. This system is simple and, as it requires little machinery, is comparatively cheap.

(2) **Treatment by decantation.**—In this system the slime is collected in a tank, where, after agitation, either mechanical or by means of compressed air, it is allowed to settle, and the supernatant liquid is decanted off; the tank is again filled up with new solution; the slime is thoroughly mixed with this by a second agitation; again allowed to settle, and the supernatant liquid again decanted off. These operations are repeated until it is found, by assaying samples taken at regular periods, that a sufficiently complete extraction has been obtained. The treated slime may then be discharged and the tank used for treatment of a new charge.

This treatment was the first employed for the treatment of slime by cyanidation, but at present is seldom employed, except at small mines which cannot afford the installation of modern machinery. A modified form of this treatment, known as semi-decantation and filtering, has replaced the old decantation process in the majority of large mills, as it requires a smaller installation for the same capacity, and yields a higher extraction in a shorter time.

(3) **Treatment by mechanical agitation and filtering** sometimes combined with semi-decantation.

(4) **Treatment by pneumatic agitation in high tanks,** followed by filtration.

(5) **Treatment by cyanidation in amalgamation pans,** without change of machinery.

The practical operation of these systems is fully described in chapters III to XV, inclusive. In addition to the above, various other systems have been proposed, but as they have not yet attained general use, it will be sufficient to simply mention them. Such systems are:

(6) **The system of Pelatan-Clerici,** of solution of the gold and silver and electric precipitation in the same tank, and

(7) **The system of direct treatment in filters** without the use of tanks.

In deciding upon the system best adapted to any particular ore, one may be guided, to a certain extent, by the practice in other mills in the same district, treating the same class of ore; but in order to be certain that the system chosen is the one best adapted to the ore in question, it would be well to consult a competent metallurgical engineer.

CHAPTER III.

CYANIDE PRACTICE OF THE EL ORO MINING & RAILWAY CO.,

El Oro, Mexico.

The following description of the accompanying flow-sheet (Fig. 1), which was kindly furnished the author by A. F. Main, general manager for the company, gives the practice in the El Oro plant during July, 1910.

Crushing and sand treatment.—The mine ore is an oxidized quartz carrying a small amount of pyrite and has an average gold value of $8.06 with 3 oz. of silver per ton. The ore after passing Comet D crushers, set at 2 in., is conveyed to the 4000-ton mill-bin. The mill consists of 10 ten-stamp batteries, the stamps weighing 1100 pounds, with an 8-in. drop, 102 per minute. The stamp duty for actual running time is 10.33 tons per stamp per 24 hours. The screens are set with a 3-in. discharge, with the screens equally divided between 6 and 8-mesh. The ore is crushed in cyanide solution containing 0.039% of KCN and 2 lb. of CaO per ton of solution. The batteries use 16 gallons of solution per stamp per minute, or about 9 tons of solution per ton of ore.

The pulp from the stamps goes to 10 cone separators, each 4.5 ft. in diameter. The slime-overflow goes direct to the preliminary-treatment tanks and the sand-underflow goes to 9 tube-mills, each mill being provided with a 3-ft. thickening-cone. The overflow from the thickening-cones joins the discharge from the tube-mills and runs to eight 7.5-ft. cones, used for final separation and which are placed below the tube-mills.

The overflow from the lower cones joins the main launder to the preliminary-treatment tanks and the underflow goes to a 40-ft. tailing-wheel which elevates 600 gallons of fine sand and water per minute, and delivers same to a distributing-box. The box is so arranged that the fine sand is delivered to three tube-mills, which are set apart for regrinding only, or to any or all of the other nine mills. The discharge from the three regrinding-mills is sent to the same 7.5-ft cones which receive the discharge from the other nine mills.

Slime treatment.—There are 15 tanks for preliminary treatment of the slime, each 34 ft. in diameter by 12 ft. deep, each having a capacity of from 100 to 150 tons of dry slime, together with the necessary solution. From the time of leaving the batteries until

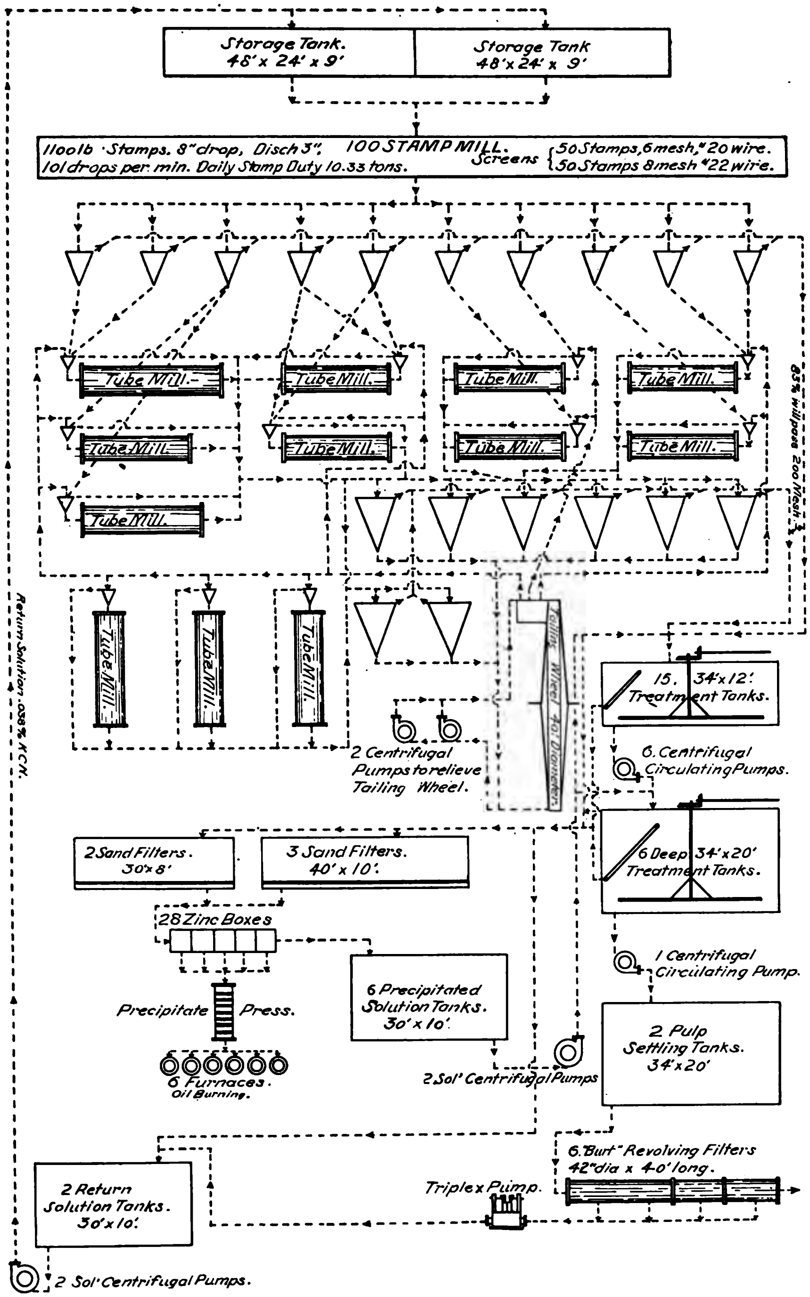


Fig. 1. Flow Sheet of the El Oro Mill.

the slime reaches the tank there has been dissolved about 80% of the gold and 13% of the silver contents, and the ore has been reduced to such a fineness that 83% of it will pass a 200-mesh screen. The current of slime is divided between three tanks where the slime settles and the solution overflows to the return-water tanks. As soon as a tank is charged, which takes about nine hours, the slime current is diverted into another tank and the first one allowed to settle from three to five hours. The clear solution is then decanted to the solution filters. Agitation is started and is accomplished by means of mechanical agitators and at the same time the pulp is circulated by a centrifugal pump, which has its suction at the bottom of the tank, and which discharges to the surface of the charge. The pump is kept running for one-half hour and then a sample is taken for determining the specific gravity of the pulp and the weight of the charge.

The pump is then stopped and a first wash put on from the barren solution with the addition of 15 kg. of NaCN and 15 kg. of lead acetate. The cyanide and acetate are suspended in a bucket, made from coarse screen, which is hung just below the surface of the circulating pulp so that the contents are dissolved slowly. Agitation continues for eight hours, during which time a small quantity of air is introduced through a pipe having a chilled steel cap with a hole of 3/16 in. diameter. The agitation is helped greatly by having some 2 by 12 in. plank placed on the inside of the tank at an angle of about 45 degrees from the horizontal. The 2-in. edge of the plank is bolted to the side of the tank and the 12-in. surface extends out into the tank. When the pulp is in agitation this causes the heavier particles to climb up the sloping board from the bottom and to be thrown toward the center of the tank where they are kept in motion by the agitator-arms. After agitating for the required length of time the charge is allowed to settle from four to six hours and the clear solution decanted to the solution filters.

The second wash is then put on from the precipitated solution and the charge agitated for another two hours, settled and decanted. The slime is then pumped by means of centrifugal pumps to one of the final treatment tanks, of which there are six, 34 ft. in diameter by 20 ft. deep. These tanks hold two of the charges from the preliminary treatment tanks, which are so timed that two will be ready to send to the final treatment tanks at the same time. Here one, and if time admits, two washes of barren solution are given and the charge is agitated for two hours with the usual time for settling and decanting. From these tanks the slime is pumped to one of two thickening-tanks 34 ft. in diameter by 20 ft. deep. These tanks are provided with overflow launders and slow agitators making 2½ revolutions per minute, so that the pulp as delivered to the six Burt revolving-filters for final dewatering and washing to recover the soluble values, cyanide and lime, contains only 40% moisture. The filters reduce the moisture to about 25%. The slime cake is

washed with barren solution and then with water before discharging to the waste dump.

Precipitation.—The richest solution in the plant is the mill solution, so a portion of it is precipitated daily and replaced with precipitated solution from the sumps below the zinc-room. This mill solution and all decanted solution is clarified in five solution-filters with sand bottoms. The precipitation room contains 28 five-compartment zinc-boxes and these handle all of the solution from the solution filters. All compartments are kept full of zinc as with a heavy flow it has been found necessary to do so in order to avoid tailing of high value. All of the zinc-boxes are cleaned each week, half on Monday and half on Tuesday. The precipitate is sluiced from the boxes into launders which deliver it, together with a small amount of short-zinc, to a 60-mesh screen placed over an iron sump 4 ft. by 6 ft. by 6 ft. Here the precipitate is washed through the screen by means of a hose, and the short-zinc which stays on the screen is recovered and returned to the head compartments of the precipitation-boxes.

A triplex pump lifts the precipitate to a 24-frame filter-press which forms a 2½ by 21 by 21 in. cake. After filling the press, steam is turned on for an hour and then air for one-half hour, after which the press is discharged into a steel, steam-jacketed drying car. The car is locked up in a strong-room and the precipitate allowed to dry until it contains about 20% moisture. It is then weighed and fluxed with 9% borax, 6% soda, 3% lime and 10% old assay slag. This mixture is then briquetted in a press, the briquettes being about 3 inches in diameter by 3½ in. long. These are charged into the crucibles without further drying.

The precipitate is melted in six oil-burning furnaces. The cost of fuel-oil is slightly higher than other fuels, but it is more economical in the long run, as a higher heat is obtained, a more liquid slag made, and less time needed for melting. The percentage of assay values contained in the original ore actually recovered as bullion represents 94.18% of the gold and 78.56% of the silver, or 91.41% of the value in dollars. The cost of operation of this company per ton of ore during June, 1910, was as follows:

Mining	$1.66
Development	.72
Milling	.17
Cyaniding	.74
Water Supply	.01
General Expenses	.22
Taxes	.28
Total	$3.80

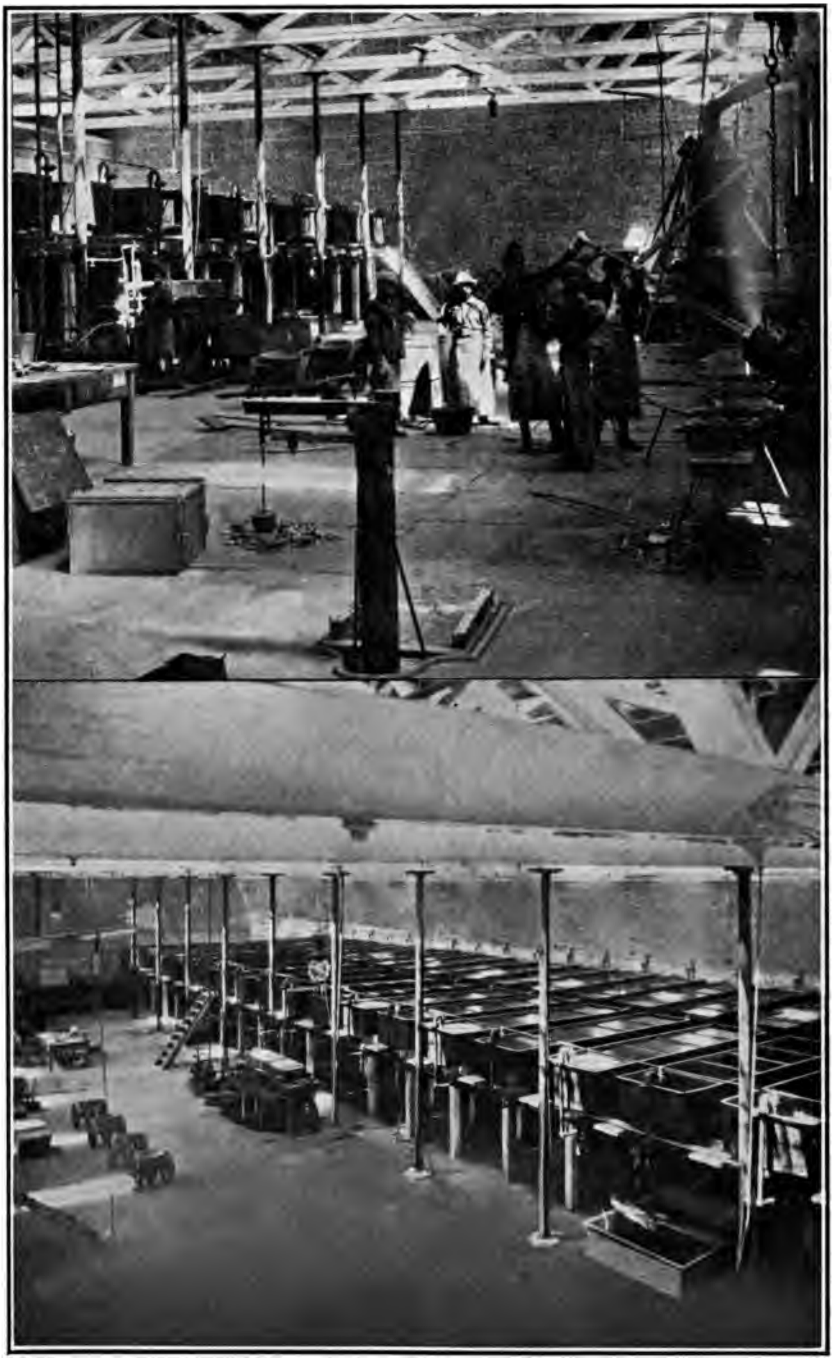


Fig. 2. Melting Room and Precipitating Room, El Oro Mining & Ry. Co., El Oro.

CHAPTER IV.

CYANIDATION PRACTICED IN THE MILLS OF THE DOS ESTRELLAS COMPANY,

Tlalpujahua, Michoacan.

The following was supplied to the author by Walter Neal, cyanide superintendent, by permission of Henri Bossuat, general manager for the Company, and represents the practice in these mills in January, 1909, which remained practically the same during 1910.

The ore is hauled from the mine to the mills in electric tram-cars, Westinghouse system. It is principally quartz carrying small quantities of slightly-oxidized iron pyrite. The silver occurs principally as sulphide, with a little chloride, while the gold is present both in the free state and mixed with the sulphide of silver. This company has two mills, and as the practice is slightly different in each they will be described separately.

Dos Estrellas Mill No. 1.

Details of the crushing machinery.—The ore in this mill is first crushed in 5 Blake crushers, size 9 in. by 15 in., after which it passes to the stamp batteries.

There are 130 stamps, of which 40 weigh 1000 lb. and 90 weigh 1200 lb. These stamps crush 500 metric tons of ore daily through a No. 26 slot screen, or about 3.84 tons per stamp per 24 hr., using 8.5 tons of solution, containing 0.10% KCN per ton of ore ground. The stamps drop 104 times per minute, 6-in. drop, and as the dies are placed level with the bottom of the screen the pulp discharges rapidly.

Classification of sand and slime.—The ore, after being ground in the batteries, is classified into the two products, sand and slime, by 4 cone separators, 6½ ft. in diam., having an angle of 47% at the vertex. The slime which overflows from these cones passes directly to one of the 17 slime-collecting tanks, while the sand discharge is fed into 4 tube-mills, each 24 ft. long by 5 ft. in diameter, revolving at 28 revolutions per minute. Each of these mills regrind 75 tons of sand in 24 hours. Instead of using flint pebbles in the tube-mills, as is the usual practice, a hard quartz ore carrying sufficient precious metal to pay for its extraction is employed. Each mill consumes about five tons of this ore per day, which is intro-

duced into the mill while in motion through a special worm-feed at the discharge end.

After being reground in these mills the pulp is elevated by a belt elevator to another cone-classifier 5 ft. in diameter with an angle of 53° at the vertex, whence the sand discharged at the vertex flows to another tube-mill, while the slime and fine sand overflows into another cone-classifier 7 ft. in diameter with an angle of 53° at the vertex, whence the slime overflow passes to one of the 12 slime-agitation tanks, while the fine sand discharged from the

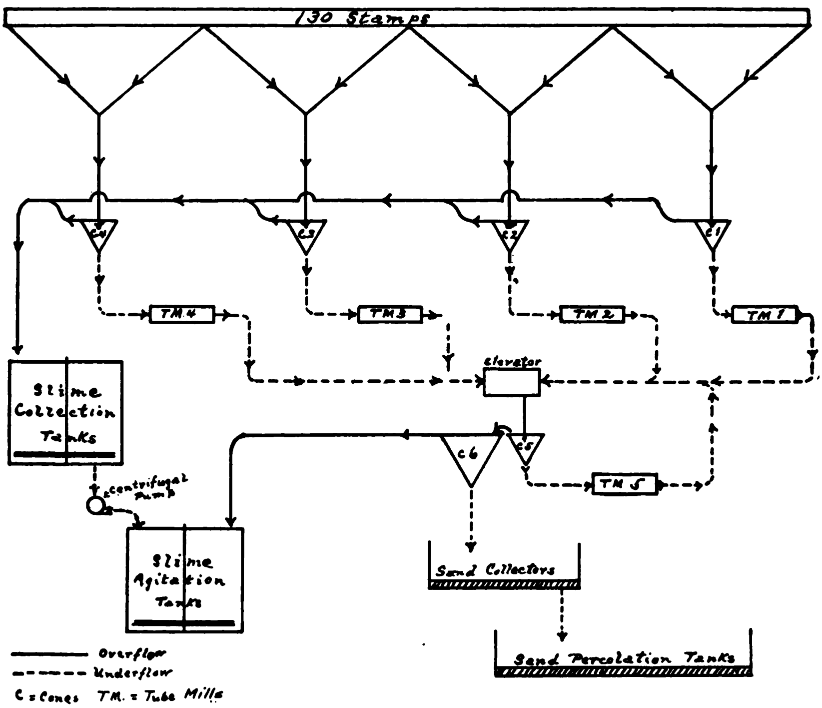


Fig. 3. Flow Sheet, Dos Estrellas Mill, No. 1.

vertex passes to one of the 10 sand-collecting tanks. By the classification just described the pulp is divided into the two products of fine sand and slime, in the proportion of 20% of the former to 80% of the latter. The products derived from the various operations of milling and classification in this mill have been classified according to size, by separating the different sized material in each product by screens, and weighing each size, and the results are shown in the following table in percentages. The figures at the head of the columns represent the mesh of the screen, and the sign + before

these figures means that the percentage of the product, indicated in the column below this figure, is larger than this screen mesh, although smaller than the preceding mesh.

TABLE No. I.

Mill products, Dos Estrellas Mill No. 1, January, 1909.

+30 %	+40 %	+80 %	+100 %	+150 %	+200 %	—200 %	
1.50	4.00	19.00	8.00	13.00	2.50	52.00	Battery pulp from No. 26 slot screen.
....	.....	1.00	1.50	12.00	3.00	82.50	Slime entering the slime-collection tanks.
....	.....	.....	.....	0.50	1.00	98.50	Overflow from the slime-collection tanks.
....	.....	5.00	5.50	21.00	4.00	64.50	Overflow from Cone No. 5.
....	.....	3.50	4.00	21.50	2.00	69.00	Overflow from Cone No. 6.
1.50	34.00	34.50	7.50	17.00	5.00	29.50	Discharge from vertex of Cone No. 1. (Heads tube-mill No. 1.)
2.00	10.50	30.00	9.50	17.50	2.00	28.50	Discharge from vertex of Cone No. 2. (Heads tube-mill No. 2.)
2.00	11.00	31.50	8.50	18.50	3.00	25.50	Discharge from vertex of Cone No. 3. (Heads tube-mill No. 3.)
2.50	11.50	32.00	10.00	18.50	3.00	23.50	Discharge from vertex of Cone No. 4. (Heads tube-mill No. 4.)
1.50	8.00	30.00	8.00	19.50	2.50	30.50	Discharge from vertex of Cone No. 5. (Heads tube-mill No. 5.)
....	1.50	12.00	8.00	18.00	2.50	58.00	Discharge from vertex of cone. No. 6. (To sand collecting-tanks.)
....	.....	2.50	3.50	17.50	8.00	68.50	Tailing from tube-mill No. 1.
....	.....	11.00	7.00	24.50	9.00	58.50	Tailing from tube-mill No. 2.
....	.....	9.00	9.50	22.50	2.00	57.00	Tailing from tube-mill No. 3.
....	.....	6.50	6.50	21.50	4.00	61.50	Tailing from tube-mill No. 4.
....	.....	13.50	9.00	20.50	3.50	53.50	Tailing from tube-mill No. 5.
....	.....	.....	14.50	35.50	3.50	46.50	Sand in sand-collecting tanks.
....	.....	.....	.....	4.50	3.00	92.50	Overflow from sand-collecting tanks.
....	.....	9.50	11.00	22.50	3.00	54.00	Pulp entering the belt elevator.

Specific gravity determinations.	Sp. gr.	Per cent Solution
Battery Pulp	1.079	88
Pulp entering belt-elevator	1.242	67
Tailing from tube mill No. 2	1.342	57
Tailing from tube mill No. 3	1.376	54
Tailing from tube mill No. 4	1.403	52
Tailing from tube mill No. 5	1.255	66

Sand Treatment—There are 10 sand-collecting-tanks, each 21 ft. in diameter bv 5½ ft. deep. These dimensions correspond to a total capacity of 22,860 cu. ft., or 228 cu. ft. capacity for each ton of sand resulting from the classification every 24 hours. The sand collected in one of these tanks remains in it for 95 hours, during which time it is treated with 50 tons of 0.27% KCN solution. The extraction obtained during the grinding and collection is 40.6% of the silver and 66.5% of the gold. From the collecting tanks the sands are transferred by cars to the lixiviation tanks. There are 17 of these tanks each 32 ft. in diameter by 4½ ft. deep, with filter bottoms, which corresponds to a total capacity of 61,526 cu. ft., or 615 cu. ft.

of capacity per ton of sand treated per 24 hours. In these tanks the sand is given 4 washes with strong solution, containing 0.4% KCN, at intervals of 4 hours. Then 4 washes with 0.27% KCN solution, at intervals of 4 hours, and finally 63 washes with weak, 0.13% KCN solution, at intervals of 2 hours. These washes give a total of 5.3 tons of solution per ton of sand. The total time during which the sand remains in the lixiviation tanks is 8 days 18 hours. The total extraction obtained by the treatment is 67.8% of the silver and 89.5% of the gold contents.

Slime treatment.—The slime collected in one of the 17 slime-collection tanks is allowed to settle and accumulate until there is a sufficient quantity for treatment, the supernatant solution which is decanted off during this operation is elevated by pumps, part going to the precipitation-boxes and part to the stock-tanks which supply the battery with solution. As soon as a sufficient quantity of slime has accumulated in a tank for treatment, that is when the proportion of dry slime to solution is as 1 to 3, the slime is agitated by means of mechanical agitators as well as by compressed air, introduced through a hose and pipe moved to every part of the tank. The charge is then transferred to one of the 12 sheet steel tanks, where it is again agitated by mechanical agitators, and thence the charge is transferred to one of the 10 wooden tanks which are 30 ft. in diameter by 13 ft. deep, where a further agitation is given by mechanical agitators, performing this transference by means of centrifugal pumps so arranged as to admit air at the suction for the aeration of the charge during transference. From the wooden tanks the treated charge of slime is discharged into two Burt filters, whence the tailing is discharged into the river. The total time of treatment given to the slime is 146 hours, of which 27.4 hours is spent in agitation. Each agitation lasts for 3 hours, after which the slime is allowed to settle for 5 hours, the supernatant solution is then decanted off and pumped to 2 solution sump-tanks 21 ft. in diam. by 5½ ft. deep, whence it flows to the precipitation-boxes, the tank is refilled with new solution, and the next series of agitation, decantation, etc., is immediately performed. The solution used in the treatment of the slime contains 0.13% KCN. The proportion of dry slime to the total solution used during the treatment is as 1 to 12. The total extraction during the treatment is 70.8% of the silver and 92.0% of the gold. In this mill it is the custom to add lead acetate to the cone-separators above the tube-mills.

Solution tanks.—There are 3 tanks for working solutions in this mill, each 36 ft. in diameter by 20 ft. deep, which have a capacity of 61,073 cu. ft or 122 cu. ft of solution per ton of ore treated. There are 3 sump-tanks 20 ft. in diameter by 6 ft. deep, and 2 of 30 ft. in diameter by 8 ft. deep, which give a total capacity of 16,948 cu. ft. or 34 cu. ft. capacity for every ton of ore treated.

There are also 5 filter-tanks 30 ft. in diameter by 8 ft. deep, for clarifying the solutions before running them through the precipitation-boxes.

Precipitation and melting.—The precipitation in this mill is performed by passing the solution through boxes divided into 6 compartments, in which the first compartment is filled with trays, one on top of another, containing zinc shorts, while the other compartments contain zinc shavings. The precipitation boxes in this mill have a capacity of 2,412 cu. ft. of solution or of 1,950 cu. ft. of zinc-shavings which is equivalent to 3.9 cu. ft. of zinc-shavings per ton of ore treated or 0.73 cu. ft. of zinc-shavings per ton of solution precipitated daily. There are 6,100 tons of solution in circulation in the mill, or 12.2 tons of solution per each ton of milling capacity. Of this solution 3,000 tons are precipitated every 24 hours, which represents 1.53 tons per cu. ft. of zinc in the precipitation-boxes. The strong solution on entering the precipitation-boxes assays 38. gm. of silver and 4.8 gm. of gold per ton, and the weak solution assays 16 gm. of silver and 1.7 gm. of gold per ton, while after passing through the boxes it is found that 95% of its silver and 92% of its gold contents have been precipitated. The precipitated slime is melted without any preliminary treatment and yields 60% of its weight in bars having a fineness of 846, of which fineness 762 represents the silver and 84 the gold content.

In the *Mining and Scientific Press* of February 27, 1909, Walter Neal published an article on the treatment of the silver and gold precipitates of this company, from which the following extract is taken:

The precipitated slime discharged from the zinc-boxes first passes through a 20-mesh screen, then through a 60-mesh screen and thence into the first cement sump-tank. The zinc shorts which remain on the 20-mesh screen, after being well scrubbed with brooms, are placed in trays and returned to the first compartment of the precipitation-boxes. The shorts which pass through the 20-mesh screen but remain on the 60-mesh screen are dried and melted after mixing with a special flux, without any acid treatment. These shorts carry from 5% to 10% of silver and gold, and the flux used is the following:

Zinc shorts	100
Borax-glass	40
Sodium bicarbonate	20
Quartz sand	10
Burnt lime	5

This flux produces a very liquid slag containing about 40% of zinc and very little silver and gold. The bars resulting from this melt assay 800 fine in silver and gold and 200 in zinc. Their weight being very little in comparison with the total weight of the bars produced in each clean-up, they are remelted together with the other bars before shipment.

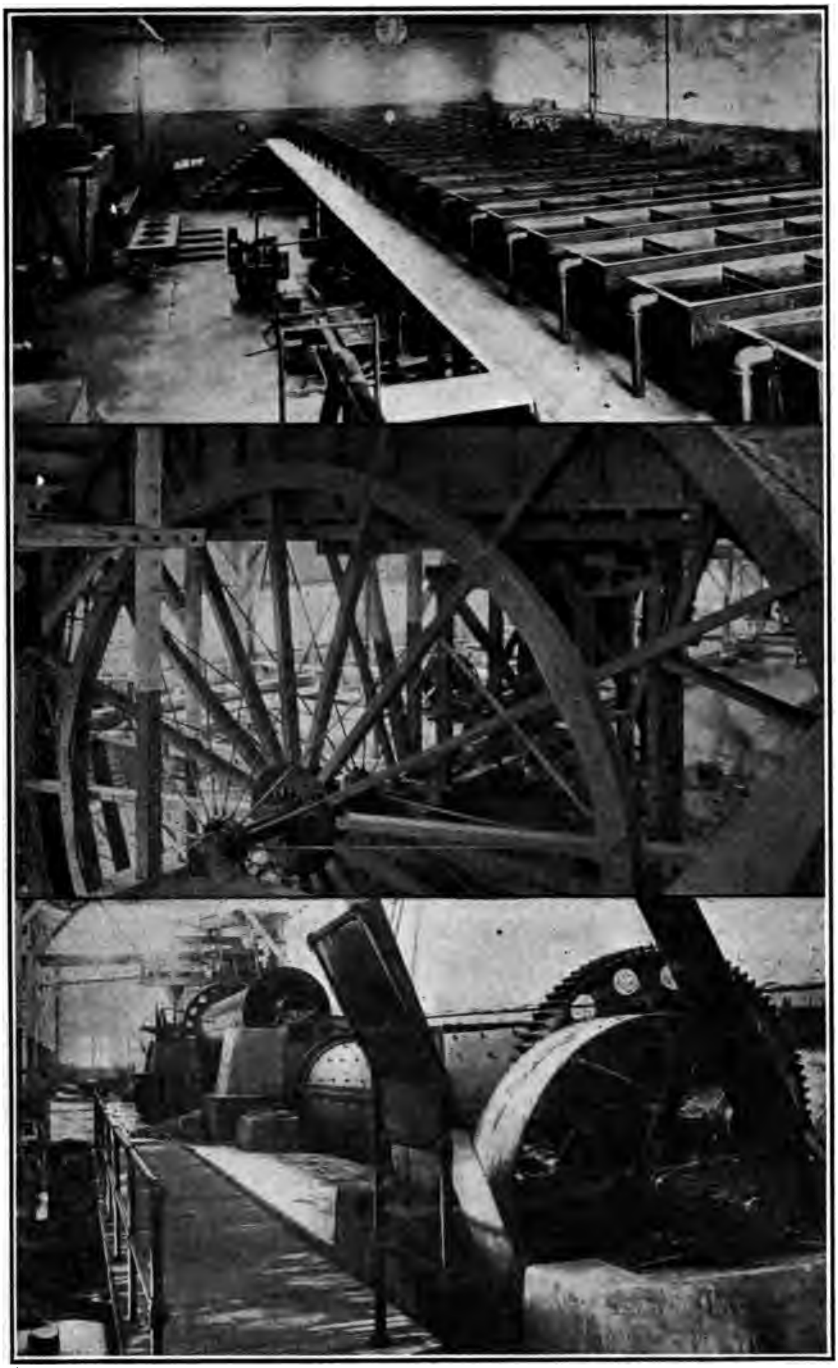


Fig. 4. Precipitating and Melting Room, Dos Estrellas Mill, No. 1; Tailing-wheel, Dos Estrellas Mill, No. 2; Tube-mills, Mill No. 2.

From the first cement sump-tank the precipitated slime overflows into the second sump-tank, from which it is forced through the filter-press by a triplex pump. The object in having two sump-tanks is that the greater portion of the precipitate may remain in the first tank, so that the pump, which forces the contents of this tank through the filter-press, may work at full speed without any attention until the clean-up of the precipitation-boxes is completed, as the quantity of precipitate carried over into the second tank is not sufficient to fill the filter-press. As soon as the clean-up of the precipitation-boxes is finished the pumping from the second sump-tank is continued until that tank is emptied, while at the same time the launders and zinc 'shorts' are washed. The precipitate from the first sump-tank is then pumped through the press, special care being given to the press during this operation to prevent a continuance of the pumping after the press is full. The two sump-tanks are then washed, and all washings passed through the press. Upon the conclusion of these operations the filter-cake in the press is partially dried by passing compressed air through it. The press is then discharged by allowing the filter-cakes to fall into a steam-jacketed car placed below the press. As soon as this car is full or has received all of the precipitates, it is run out from under the press, connected to a steam pipe, and allowed to stand all night with steam entering the jackets. The steam required for this purpose is furnished by a small vertical boiler, in which it is found that by firing for a couple of hours sufficient steam is generated to last all night and to dry the precipitate so that it contains but 18% of moisture, in which state it is ready for fluxing and melting. Formerly the precipitate was fluxed and made into briquettes before melting, but this practice has been discontinued, as it is believed that the numerous handlings of the precipitate caused considerable loss. The present practice requires but two handlings of the precipitate as against six handlings in the briquetting process, as shown below:

Handlings in Direct Melt.

1. The filter cakes are detached from the press forms, and allowed to fall into the drying-car, being careful not to break them up more than absolutely necessary. The car is moved to the place where it may be connected with the steam pipe and dried. The car is then weighed, to determine the weight of the precipitate contained, and consequently the amount of flux required. A uniform layer of the flux is then spread over the precipitate in the car without mixing, and the car is pushed to the side of the furnace.
2. The precipitate and flux is then shoveled into the crucibles.

Handling in Briquetting and Melting.

1. The filter-cakes are detached from the filter and fall into the drying-car, which is pushed to the steam-connection.

2. The dried cakes are shoveled from the car into boxes, where they are weighed and mixed with flux.

3. The precipitate and flux are mixed together in the box by shoveling, and the boxes are carried to the briquetting-machine.

4. The precipitate and flux are shoveled from the box to the hopper of the briquetting-machine.

5. The briquettes are lifted by hand and placed in a box to carry them to the side of the furnace.

6. The briquettes are placed in shovels and thrown into the crucibles.

There is no doubt but that there are advantages in briquetting the precipitate, and that it is well to thoroughly mix the precipitate and flux, but the loss incurred through many manipulations may be appreciated by anyone who has touched the precipitate with the finger and afterwards tried to clean it on a handkerchief. Moreover the necessity for a thorough mixing of precipitate and flux is doubtful, as in this mill it appears that the precipitate melts equally well, and gives as clean slags by placing the flux on top of the precipitate in the car, without mixing, as when thoroughly mixed in briquetting. If care is taken not to break the filter-cakes which fall into the drying car, the formation of a crust on the surface of the mass in the crucible upon melting, which may cause losses upon the escape of the steam enclosed within it, does not occur more often than when melting briquettes. There is little loss in dust, as the precipitate introduced into the crucibles is sufficiently moist to prevent such loss.

The following flux is used in melting the precipitate slime:

	Parts
Precipitate	100
Borax-glass	15
Sodium bicarbonate	8
Quartz sand	4
Pieces of wrought iron in excess.	

The precipitate yields from 60% to 80% of its weight in bars. No. 400 Dixon graphite crucibles are used, which hold 86 kg. of precipitate and flux. Coke furnaces are used for melting. As soon as the precipitate has melted the crucible is removed from the furnace and the upper portion of its contents, consisting of the slag, is poured into a conical slag pot, having a tapping hole three inches above the bottom, plugged up with clay. The rest of the contents of the crucible, consisting of bullion with a little slag, is then poured into the bar molds.

As soon as the slag in the slag pot has cooled so as to form a crust ½ inch thick on the sides, the clay plug is removed from the tapping-hole, and the liquid slag in the interior of the pot above the tapping-hole is allowed to run out into molds, taking a sample

of the slag as it runs, and granulating this sample by dropping in water, so as to be able to assay and calculate the value of the slag, which is then sold to the ore-buyers.

The crusts which remain in the conical molds, together with the slag on top of the bars in the metal molds, are remelted, and poured into conical molds in order to draw off the poor slag through the tap-hole as before. The slabs of bullion from the metal molds, with the buttons of bullion from the bottom of the slag-molds, and the bars obtained in melting the zinc 'shorts' previously mentioned, are all remelted together. Any slag which they may contain is removed with an iron spiral, and the molten metal is poured into molds without undergoing any refining. A sample is taken from the molten bullion after removing the slag, and another sample is taken while pouring, and it is interesting to note that the difference between the results of the two assays during the year, while melting 35,230 kg. of gold and silver was only $1,963.69, the sample taken during pouring being the higher.

Chemical consumption.—In this mill the consumption of chemicals per ton of ore treated is as follows:

Potassium cyanide	678 gm.
Zinc	576 "
Lead acetate	144 "
Lime	14 kg.

Cost of treatment.—The cost of treatment in this mill per ton of ore is as follows:

Milling.

Cost of milling in crushers and batteries	₱0.848	
Cost of milling in tube mills	.312	
Total cost of milling		₱1.16

Cyanidation.

Cost of labor	₱0.5065	
Cost of merchandise	1.1225	
Cost of electric power and light	.3015	
Total cost of cyanidation		1.9305
Total cost of milling and cyanidation		₱3.0905

Dos Estrellas Mill No. 2.

Crushing.—This mill, also known as the Cedro, has six Blake crushers, 8 in. by 15 in., in which the ore is crushed before feeding to the stamp batteries. There are 120 stamps, each weighing 1250 lb., which have 104 drops per minute, 6 in. drop; the ore is crushed through a No. 26 slot wire screen, set at the level of the dies, using 10 tons of 0.12% KCN solution per ton of ore crushed. These batteries grind at the rate of 4.16 tons of ore per stamp, or 500 tons of ore per 24 hours.

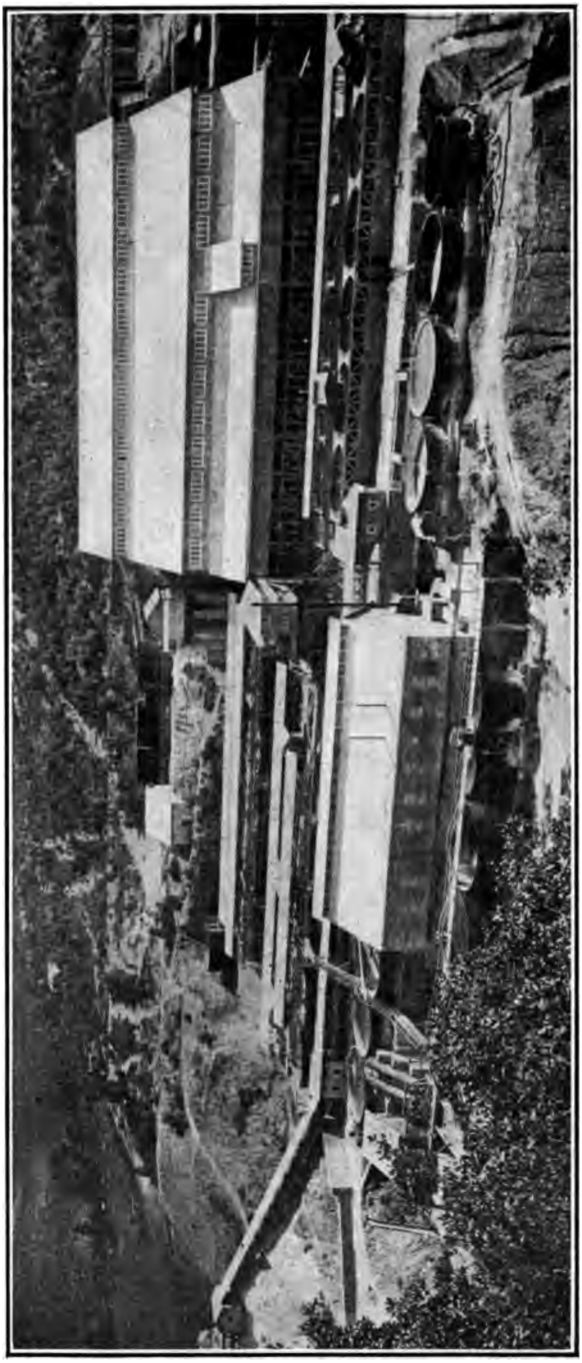


Fig. 5. General View of Dos Estrellas Mill, No. 2.

Classification of sand and slime.—The ore, after being ground in the batteries, is separated into sand and slime by cone-classifiers, the sand is reground in tube-mills, and again classified, until the whole of the pulp is converted into the two products of fine sand and slime, the former consisting of 30% and the latter 70% of the total weight.

This classification is shown graphically in the diagram, Figure No. 6.

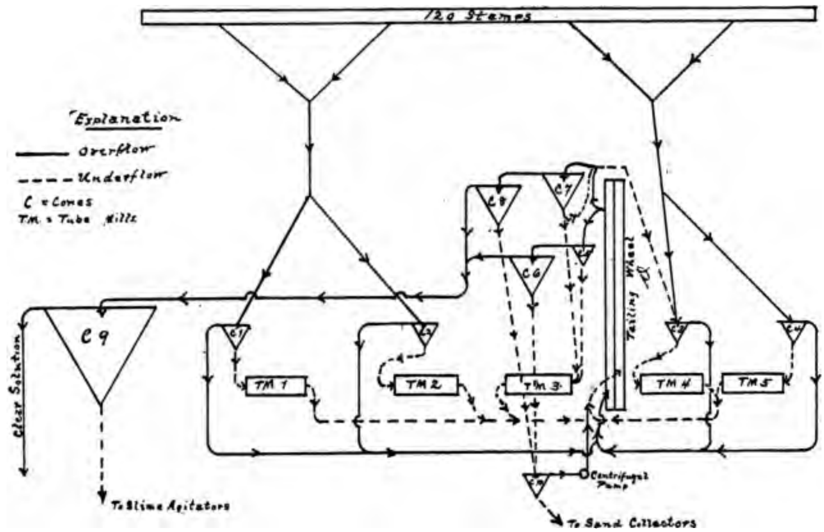


Fig. 6. Flow Sheet, Dos Estrellas Mill, No. 2.

In the above diagram the cones C1, C2, C3 and C4 are each 7 ft. in diameter with an angle of 50° at the vertex, the cone C5 is 5 ft. 8 in. in diameter with an angle of 47° at the vertex, the cones C6, C7 and C8 are each 8 ft. in diameter with an angle of 42° at the vertex, and the cone C9 is 34 ft. in diameter by 35 ft. deep, having an angle of 53° at the vertex.

Explanation of the Diagram.

The batteries are divided into four sets of 30 stamps each. The pulp from the first two sets flows to cones, C1 and C2, while the pulp from the second two sets of batteries flows to cones C3 and C4. The overflow from these four cones passes to the slime-compartment of the tailing-wheel, while the sand discharged from the vertices are fed to the four tube-mills, 1, 2, 4 and 5, for regrinding, and thence pass to the sand-compartment of the tailing-wheel, by which they are elevated, the slime being delivered to cone C7 and the sand to cone C5. The sand discharged from the vertices of

these two cones is fed to the tube-mill No. 3, whence, after regrinding, it flows to the tailing-wheel for elevation, while the slime which overflows from these two cones flows respectively into cones C8 and C6, whence the fine sand discharged from the vertices flows to the dewatering-cone, C10, while the slime overflows into the dewatering-cone C9.

From the cone C10 the fine sand discharged from the vertex flows to the sand-collection-tanks, while the overflow is returned to the sand-compartment of the tailing-wheel by a centrifugal pump. From the cone C9 the thickened slime discharged from the vertex flows to the slime-agitation-tanks, while the clear solution overflowing is returned to the solution feeding the batteries, or is precipitated as may be required. The five tube-mills are each 5 ft. in diameter by 24 ft. long. The result of the milling and classification above described is shown in the following table of sizing tests of the pulp during different stages of the operations just described.

TABLE No. II.

Mill products of Dos Estrellas Mill No. 2, December 31, 1908:

+30	+40	+80	+100	+150	+200	—200	
7.50	10.00	17.00	6.00	12.50	2.50	44.50	Battery pulp through 16 by 26 mesh slot screen.
10.50	14.00	30.50	8.00	15.50	5.00	16.50	Heads of tube mill No. 1.
2.00	2.50	23.50	11.50	28.50	4.50	27.50	Heads of tube mill No. 2.
7.50	6.00	31.00	13.00	24.00	3.00	15.50	Heads of tube mill No. 4.
14.50	15.00	31.00	7.00	16.00	2.00	14.50	Heads of tube mill No. 5.
....	.....	.....	1.50	7.00	5.00	86.50	Overflow from Cone C1.
....	.....	0.50	1.00	5.50	3.50	89.50	Overflow from Cone C2.
....	.....	.....	.....	3.50	0.50	96.00	Overflow from Cone C3.
....	.....	.....	0.50	3.50	1.50	94.50	Overflow from Cone C4.
....	.....	18.50	8.50	22.50	5.00	45.50	Tailing of tube mill No. 1.
....	.....	18.50	6.00	29.00	8.00	46.50	Tailing of tube mill No. 2.
....	.....	13.50	13.00	27.50	6.00	40.00	Tailing of tube mill No. 3.
....	.....	16.00	14.00	26.50	5.00	38.50	Tailing of tube mill No. 4.
.....	1.50	16.00	8.00	24.00	6.50	44.00	Tailing of tube mill No. 5.
....	.....	21.00	15.50	26.00	5.00	32.50	Discharge from vertex of Cone C5. (Heads tube-mill No. 3.)
....	.....	20.50	13.50	25.50	5.00	35.50	Discharge from vertex of Cone C7. (Heads tube-mill No. 3.)
....	.....	4.50	8.50	27.50	7.00	52.50	Discharge from vertex of Cone C6 and C8.
.....	0.50	26.00	13.50	24.50	4.00	31.50	Discharge from vertex of Cone C5 (to Cone C2).
....	.....	27.50	17.00	27.00	4.50	24.00	Discharge from vertex of Cone C7 (to Cone C3).
....	.....	4.00	5.00	17.00	4.00	70.00	Overflow from Cone C5.
....	.....	1.50	2.00	19.50	6.00	71.00	Overflow from Cone C7.
....	.....	.....	.....	6.00	2.00	92.00	Overflow from Cone C6.
....	.....	.....	1.00	10.50	3.50	85.00	Overflow from Cone C8.
....	.....	.....	.....	6.50	4.50	89.00	Overflow from sand collecting-tanks.
....	.....	20.50	10.50	31.00	5.50	32.50	Sand in sand collecting-tanks.
....	.....	.....	.....	7.50	4.50	88.00	Slime discharged from Cone C9.
....	.....	17.50	1.00	23.00	6.00	52.50	Discharge from tailing-wheel.

Specific Gravity Determinations.

	Sp. gr.	Per cent Solution
Slime flowing to the dewatering-cone C9	1.060	91
Discharge from the tailing wheel	1.111	83
General sample of tailing from tube-mills	1.429	50

The dewatering-cone C9 receives about 400 tons of slime every 24 hours diluted in the proportion of 10 of solution to 1 of dry slime, while the thickened slime discharged from the vertex has a dilution of 1 to 3.5.

The tailing-wheel is shown in (the middle plate of) Figure No. 4. It is 48 ft. in diameter, and works more satisfactorily than the centrifugal pumps and belt-elevators which were formerly used for the purpose of elevating sand and slime. The lime and lead acetate which may be required for the proper treatment of the ore are added to the cones C1, C2, C3, and C4.

Extraction in the battery.—The ore ground in this mill assays 107.9 gm. of silver and 11.29 gm. of gold per metric ton, on an average, and as it is ground in solution it is found that 17.7% of the silver and 33.3% of the gold is dissolved in the battery.

Sand treatment.—There are 8 sand-collecting-tanks, each 22 ft. in diameter by 7 ft. deep, which represents a total capacity of 21,287 cu. ft., or 122 cu. ft. per ton of sand derived from the classification every 24 hours. Ordinarily it requires 51 hours to fill one of the collecting-tanks and the extraction during grinding and collection is 27.4% of the silver and 57.8% of the gold. The sand is not given any washes in the collecting-tanks, but as soon as a tank is full the sand is transferred to a treatment-tank. There are 12 sand-treatment-tanks, each 36 ft. in diameter by 5 ft. 6 in. deep. They have a false bottom of perforated boards covered by jute sacking, and their total capacity is 67,116 cu. ft., or 383 cu. ft. per ton of ore treated. In these tanks the sand is treated with 12 washes of strong solution, containing 0.4% KCN, at intervals of four hours between washes, and afterwards with 67 washes of weak solution, containing 0.12% KCN, at intervals of two hours between washes. The duration of the sand treatment is 7 days 18 hours. The proportion of sand and solution is as 1 to 2.4. The total extraction obtained during the sand treatment is 57.1% of the silver and 90.5% of the gold. The sand is discharged from the tanks by a Blaisdell excavator discharging on a belt-conveyor, which carries the sand tailing to the left of the mill, as shown in the general view, Figure No. 5.

Slime-treatment.—There are 12 slime-collecting-tanks of steel, each 36 ft. in diameter by 10 ft. deep. These tanks have a total capacity of 122,145 cu. ft. or 325 cu. ft. per ton of slime resulting from the classification. In these tanks the slime is given six washes with 0.12% KCN solution at intervals of 8 hours. During each wash three hours are consumed in agitation and five hours in settling and

decantation. The total time of treatment in these tanks is 70 hours, of which 18 hours is devoted to mechanical agitation. After this treatment the slime is discharged into the first of a series of six deep agitation tanks, each 36 ft. in diameter by 20 ft. deep, fitted with mechanical agitators, and so arranged that the slime passes from one to the next throughout the series in a continuous current, thus obtaining continuous agitation during three days 13 hours, before the slime passes into the tanks which charge the Burt filters. In every other tank of this series compressed air is introduced to prevent the settling of the heavier particles below the stirring arms. The proportion of slime to solution during the whole slime treatment is as 1 to 11.7. The extraction obtained during the slime treatment is 68.2% of the silver and 92.7% of the gold.

Solution-tanks.—In this mill there are four tanks for working-solutions, each 36 ft. in diameter by 16 ft. deep, and two tanks 22 ft. in diameter by 7 ft. deep, giving a total capacity of 66,393 cu. ft., or 132 cu. ft. of capacity of working-solution per ton of ore treated every 24 hours. For settling solutions there are 2 tanks 36 ft. in diameter by 16 ft. deep, 3 tanks 36 ft. in diameter by 10 ft. deep, and 2 filter tanks 36 ft. in diameter by 8 ft. deep, representing together a total capacity of 77,360 cu. ft., or 154.7 cu. ft. per ton of ore treated every 24 hours.

There are 5 sump-tanks in this mill of which 2 are 36 ft. in diameter by 16 ft. deep, 1 is 36 ft. in diameter by 20 ft. deep, and 2 are 22 ft. in diameter by 15 ft. deep, giving a total capacity of 16,594 cu. ft. or 113.2 cu. ft. per ton of ore treated daily.

Precipitation and melting.—Zinc-shavings are used in this mill for the precipitation of the metals from solution, and in addition, the last compartment of the zinc-boxes is filled with oak charcoal, selected pieces, which are retained in the compartment by a 4-mesh iron screen placed over the charcoal to prevent the escape of pieces of charcoal into the solution where they might afterwards cause losses. The precipitation-boxes in this mill have a capacity for 4,492 cu. ft. of solution or deducting space occupied by false bottoms, for 2,808 cu. ft. of zinc-shavings, equivalent to 5.61 cu. ft. of zinc-shavings per ton of ore treated in 24 hours, or to 1.27 cu. ft. of zinc-shavings per ton of solution precipitated daily. There are 8,924 tons of solution in circulation in this mill, or 17.8 tons per ton of ore treated every 24 hours. Of this solution 3,000 tons are precipitated daily, or 1.06 tons per cu. ft. of zinc-shavings. The strong solution upon entering the zinc-boxes usually carries 18.9 gm. of silver and 3.7 gm. of gold per ton, and the weak solution usually carries 8.13 gm. of silver and 1.38 gm. of gold per ton. About 95% of the silver and 92% of the gold content is precipitated during the passage of the solutions through the zinc-boxes.

The precipitated slime is melted directly, without any preliminary treatment, using the same practice and flux, as in the No. 1

mill of this company (already explained), and yields 50% of its weight in bars assaying 701 fine in silver and 114 fine in gold.

Consumption of chemicals.—The consumption of chemicals per ton of ore treated in this mill is as follows:

Potassium cyanide	448 gm.
Zinc	498 "
Lead acetate	38 "
Lime	6.2 Kg.

Cost of treatment.—The cost of treatment in this mill, per ton of ore treated, is as follows:

Milling.

Cost of grinding in crushers and batteries	₱0.874	
Cost of grinding in tube-mills	.356	
Total cost of milling		₱1.23

Cyaniding.

Cost in labor	₱0.3384	
Cost in merchandise	.9539	
Cost in electric power and light	.1346	
Total cost of cyaniding		1.4269
Total cost of milling and cyaniding		₱92.656

CHAPTER V.

CYANIDE PRACTICE OF THE MEXICO MINES OF EL ORO, Ltd.

El Oro, Mexico.

The practice in this mill may be described as grinding in solution, all-sliming, and agitation in mechanical agitators.

The following details were kindly furnished to the author by Fergus L. Allen and Jesse Vincent, general manager and superintendent for the company, respectively.

Grinding.—The ore entering the mill is crushed in an 18-in. by 24-in. Blake crusher, from which it falls on a grizzly with 1½ in. openings. The ore passing through the grizzly falls into the battery-bin, while the larger pieces which do not pass through the grizzly are fed to a pair of rolls, 36 in. in diameter by 16 in. face, where they are crushed so that they fall into the same ore-bin. The crusher and rolls are run by a 100 hp. electric motor. The battery consists of 40 stamps, of 1140 lb. each, which drop 7½ in. at the rate of 102 drops per minute. An 8-mesh slotted screen is used on the batteries, placed so that its bottom is 2½ in. above the dies, and the ore is ground in the batteries at the rate of 6½ tons per stamp per 24 hours, or 260 tons per day, using from 9 to 10 tons of 0.04% KCN solution per ton of ore in grinding. The stamps are run by two 65 hp. electric motors, one being connected with each 20 stamps.

Regrinding and classification.—This mill is designed to convert all of the ore into slime, in order to treat it as such, in accordance with the modern practice of cyanidation, and for this purpose 6 Krupp No. 5 tube-mills are used, each moved by a separate 75 hp. electric motor, which gives 31 revolutions per minute to the mill; but it has been found that with four of these mills all of the sand produced by the 40 stamps can be thoroughly pulverized, as each tube-mill regrinds from 50 to 60 tons of sand per day to such a fineness that 90% of the product will pass through a 200-mesh screen.

The product of the batteries passes to two cone-separators 42 in. in diameter by 6 ft. deep, from which the slime overflows into other cones of the same size situated below, while the sand discharged from the vertices passes to the tube-mills, whence, after being reground it passes to the second set of cones just mentioned. The slime which overflows from these cones flows to the agitation-tanks, while the sand discharged from the vertices passes to another set

of hydraulic cones where it is further diluted with 0.04% KCN solution in order to separate any slime mixed with the sand. This slime overflows from the cones and passes to the agitation-tanks, while the sand discharged from the vertices is elevated by a tailing-wheel, 40 ft. 7 in. in diameter and again fed to the tube-mills for regrinding.

Slime treatment.—The cyanidation plant consists of 16 sheet iron tanks, 34 ft. in diameter by 12 ft. deep, arranged with mechanical agitators moved by 4 electric motors. Each tank has a capacity of 10,895 cu. ft., or 308.5 metric tons. The slime from the classification system is settled in one of the agitation-tanks, until it contains about 80 tons of dry slime, which is considered a sufficient charge. The supernatant solution is then decanted off and the tank filled up with 0.05% KCN solution; from ¼ to ½ lb. of lead acetate and 15 or 20 lb. of lime are added for each ton of ore-content, in order to give the solution an alkalinity of 0.05% CaO, and the charge is agitated for 18 hours. The charge is then allowed to settle for 12 hours, the solution is decanted off and the tank is refilled with 0.05% KCN solution, and the charge is again agitated for a period of from 3 to 12 hours, according to the amount of precious metal left in the slime. The charge is again settled, the solution decanted, etc., repeating the operation two or three times. The total time of treatment is 120 hours, of which ordinarily 36 hours are employed in agitation, and 4 or 5 washes and decantations are practiced.

During the agitation compressed air is introduced in the bottom of the tank, in order to provide the oxygen necessary for the solution of the gold and silver. As soon as the assays, taken every 12 hours, show that a satisfactory extraction has been obtained, the slime is discharged from the agitation tanks to the Burt filters, whence the filtered solutions pass to a sump tank, while the filtered and washed slime is discharged to the river bed. All of the solutions from the various decantations are clarified by passing through a Burt clarifier. The solutions are circulated throughout the mill by means of 4 centrifugal 6-in. Butters pumps, each run by a 20-hp. electric motor. These pumps are arranged for the introduction of air so that the solutions passing through them are aerated. There are 4 tanks for the working solutions each 30 ft. in diameter by 12 ft. deep.

Precipitation and melting.—Twelve tons of solution are used for each ton of dry slime during agitation, but only 5 tons of this solution are precipitated, as the rest is used, without being precipitated, for the first wash, in order to enrich the solution before precipitation. There are 9 precipitation-boxes, each having 5 compartments, measuring, 2 ft. by 3 ft. by 4 ft., with a false bottom made of wire screen, placed 6 in. above the true bottom. Each compartment contains about 18 cu. ft. of zinc-shavings, which has previously been submerged in a 2% solution of lead acetate, until the deposit of me-

tallic lead has given them a gray color. These shavings give a perfect precipitation of the metals. The solutions entering the precipitation boxes carry from \$1 to \$5 in gold and from ½ to 3 oz. in silver, while the solution leaving the boxes does not carry appreciable values. The precipitated slime is melted without previous treatment, but the zinc 'shorts' from the precipitation boxes are given a preliminary treatment with sulphuric acid before melting. The precipitated slime is passed through a filter-press. The cakes thus formed are dried, in a car having a false bottom through which steam passes, mixed with from 18% to 30% of their weight of a flux composed of soda, borax and metallic iron (when they contain sulphur) and compressed into briquettes. They are then melted in large graphite crucibles, Nos. 200 and 300, in 3 oil-burning furnaces.

The precipitate ordinarily yields from 73% to 78% of its weight in bars which average from 900 to 930 fine in gold and silver, the gold fineness being ordinarily from 65 to 85, while the silver fineness runs from 820 to 850.

Extraction.—The ore treated in this mill assays on an average 7 oz. of silver and \$12 in gold per ton of 2000 lb., or converted into the metric system, 240 gm. of silver and 20 gm. of gold per metric ton. The tailing after treatment assays from 1.2 to 1.5 oz. of silver and 60c. U. S. in gold, or 41.1 to 51.4 gm. of silver and 1 gm. of gold per metric ton. Consequently the extraction obtained is 80% of the silver and 95% of the gold contained in the original ore.

Chemical consumption.—Each ton of ore treated consumes the following chemicals: 1.5 lb. potassium cyanide; from 1. to 1.2 lb. of metallic zinc; from ¼ to ½ lb. of lead acetate, and from 15 to 20 lb. of lime.

Cost of treatment.—According to the official report of the company, the cost of treatment during June, 1908, in treating 8,131 tons of ore, was as follows: Cost of milling, per ton, \$0.30; cost of cyanidation, per ton, \$1.27; total cost of treatment, \$1.57.

CHAPTER VI.

CYANIDE PRACTICE OF THE ESPERANZA MINING CO.

El Oro, Mexico.

This mill was originally constructed for treating the ores of the mines of this company by pan-amalgamation. Afterwards the pans were taken out and wooden tanks for cyanidation substituted, the sand being treated by percolation and the slime by decantation. As the capacity of this mill was less than that of the daily production of the mine, the management wisely decided to increase the capacity by coarser grinding at the expense of the extraction, being content with an extraction of 86% of the contents, but saving the tailing for subsequent treatment. When they had about 450,000 tons of tailing the company erected a modern mill with all of the latest improvements not only for treating this, but also for the treatment of the ore as daily extracted from the mine. In this chapter the old mill as well as the modern mill are described, the former on account of the various ingenious devices therein used whereby satisfactory results were obtained with a cheap and simple plant, such as may be built at any small mine where production does not warrant the installation of an expensive modern mill.

The Old Mill.

The following details, representing the practice in this mill during the month of April, 1908, were courteously furnished the author by Wm. Howard, then cyanide superintendent, by permission of W. E. Hindry, then general manager for the company. The details regarding the assays, products, cost of treatment and percentage extraction were taken from the Directors' Report of 1907.

Ores.—Two classes of ores are extracted from the mines of the company, sulphide and oxidized ores, amounting together to about 450 tons per day. The sulphide ores after being ground, pass over concentrating tables, which remove the sulphide contents in the form of concentrate, before the pulp is classified into sand and slime for their respective treatment, whereas the oxidized ores are classified into sand and slime immediately after grinding.

Grinding, concentration and classification of the sulphide ores.—These ores assay on an average 30.46 gm. of gold and 307 gm. of silver per metric ton. After being hoisted from the mine this ore is thrown onto a grizzly made of wrought-iron strips placed ¾ in.

lb. of sulphuric acid								
0.28	[illegible]020	4.2	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	1.52
0.06	1250	[illegible]	60	[illegible]	1050	[illegible]		
[illegible]	[illegible]	[illegible]	35	[illegible]	[illegible]	1.25		
[illegible]	[illegible]	[illegible]	40	40	3000	1.38		
0.044	[illegible]	[illegible]	40	40	4000	0.50		
[illegible]	2200	[illegible]	[illegible]	[illegible]	200			
[illegible]	4000	[illegible]	[illegible]		[illegible]			
[illegible]	1380	14.2	60		420			
[illegible]	[illegible]	[illegible]	11		[illegible]			
[illegible]	320	[illegible]	[illegible]		[illegible]			
[illegible]	[illegible]	[illegible]	[illegible]		120			

apart. The ore already broken in the mine smaller than ¾ in. passes through the grizzly and falls into a bin below it, while the coarser ore is fed to a 13 by 24-in. Blake crusher working at 250 revolutions per minute, where the ore is crushed to ¾ in. size and thence falls into the same bin previously mentioned. From this bin the ore is carried to the mill-bin, from which it is fed, by Challenge feeders to thirteen 5-ft. Huntington mills. These mills giving 80 revolutions per minute, grind 24 tons per 24 hours to pass through a No. 60 slot screen, using from 2 to 3 tons of water per ton of ore.

Thence the pulp flows to a spitzkasten, from which the slime overflows to a cone-classifier situated below while the sand discharged at the vertex is divided between nine Wilfley concentrators, Size No. 5, whose pulleys revolve at 244 revolutions per minute, and give a ¾ stroke. The concentrate from these tables is ordinarily 5.58% of the weight of the original sulphide ore, and assays 1,596 gm. of silver and 258 gm. of gold per ton. The concentrate is sold to the ore-buying agencies of the smelters. The tailing from the concentrators passes to another spitzkasten, whence the slime, and fine sand overflow into a 4 ft. cone-separator while the coarse sand discharged at the vertex passes into two 5 ft. Huntington mills where it is reground to pass through a sheet-steel screen punched to 80-mesh, using 2½ tons of water per ton of dry pulp reground.

From the cone-separators above mentioned, which have received the slime and fine sand from the two spitzkasten, the slime which overflows passes to the slime-settling-tanks through a blanket-sluice or trough 30 in. wide and 64 ft. long, having a fall of ½ in. per foot, the bottom is lined with blankets which collect the concentrate contained in the slime. It is found that the concentrate collected upon washing these blankets assay 1,500 gm. of silver and 225 gm. of gold per ton. The sand discharged from the vertex of the cone passes to three gravity cone-separators 4 ft. in diameter, whence the slime that overflows passes through the same blanket-sluice already mentioned, while the sand discharged from the vertices passes to three other hydraulic cones 6 ft. in diameter with angle of 60° at the vertex.

The sand reground in the two Huntington mills flows into the three hydraulic cones after passing through blanket-sluices 17 in. wide and 55 ft. long, having a fall of ½ in. per foot, from which the concentrate recovered assays 1,200 gm. of silver and 150 gm. of gold. The slime which overflows from the three 6 ft. hydraulic cones passes through the blanket-sluice first mentioned to the slime-settling-tanks, while the sand discharged from the vertices passes through three blanket-sluices 30 ft. wide, 90 ft. long, having a fall of 0.84 in. per lineal foot to the sand-settling-tanks. The concentrate collected upon washing the blankets of these sluices assays 1,500 gm. of silver and 500 gm. of gold per ton.

Grinding and classification of oxidized ores.—These ores assay on an average 14.9 gm. of gold and 84.5 gm. of silver per metric ton. After being hoisted from the mine this ore is dumped on a grizzly 3 ft. wide by 20 ft. long, made of strips of wrought iron placed 1½ in. apart. The ore already broken smaller than 1½ in. falls through the grizzly into a bin below it, while the coarse ore is fed into a 13 in. by 24 in. Blake crusher, running at 250 revolutions per minute where it is crushed to 1½ in. and falls into the same bin. From this bin the ore is carried to the mill in cars where it is distributed among the Challenge feeders which supply the stamp batteries.

The mill contains 120 stamps, weighing 950 lb. each. The stamps give 102 drops per minute, drop 6½ in. and grind 1.87 tons per stamp per 24 hours to pass through a 40-mesh wire screen, using from 3 to 4 tons of water per ton of ore. The bottom of the screens are placed 8 in. above the dies; 6 kg. of lime is added to the battery with every ton of ore, in order to neutralize the acidity of the ore and to cause the slime to settle quickly in the subsequent treatment.

The pulp from the batteries passes into two hydraulic cone-separators for its classification into sand and slime. The slime overflow passes into the slime-settlement and treatment-tanks, while the sand discharged from the vertices flows into the sand-collection-tanks through blanket-sluices 30 in. wide, 90 ft. long, having a fall of 0.84 in. per lineal foot. The concentrate collected from these blankets on washing assays 1,500 gm. of silver and 500 gm. of gold, per ton. Any slime which may enter into the sand-tanks is kept in suspension by laborers armed with hoes who continually stir the solution and allow the slime and excess solution from the settlement of the sand to overflow into the slime-tanks. From the slime-tanks after settlement the excess water is decanted into tanks from which it is afterwards pumped to the tanks above the mill which supply the batteries and Huntington mills.

Size of the sand.—The sands resulting from the grinding and classifying of the sulphide and oxidized ores are treated together in the same tanks.

A screen analysis of the sand shows the following sizes:

	Per cent
Sand coarser than 50 mesh	4.5
Sand finer than 50 mesh but coarser than 80	22.25
Sand finer than 80 mesh but coarser than 100	40.25
Sand finer than 100 mesh	33.00

Construction of the tanks.—The sand-plant consists of 16 rectangular wooden tanks 18 ft. by 30 ft. by 4 ft., or a total of 34,560 cu. ft. of capacity, of which 192 cu. ft. are required per ton of ore per day. These tanks are arranged with a false bottom, below which are placed the pipe connections for introducing and withdrawing the water and cyanide solution used in the treatment. The false bottom

is made as follows: upon the bottom of the tank place a number of 3 by 4 in. wooden beams, 1 ft. apart. On top of and across these beams lay a floor of 1½ in. boards having numerous ¾ in. holes bored through them (say every 4 in.). On top of the boards lay a cocoa matting, the edges of which are held in place by an inch rope forced into the space left for this purpose between the edges of the boards and the sides of the tank. Attached to one side of the inside of the tank is a sort of box for the discharge of water and slime while filling the tank. This box is formed of two boards placed 1 ft. apart. These boards stand vertically and have their outer edges attached to the side of the tank in such a manner that the direction across each board forms a right angle with the side of the tank. The inner edges of the boards have guides, in which are placed narrow wooden blocks, reaching from the guides of one board to those of the other. These blocks form the interior door of the box, and by placing one on top of another, as the sand rises in the tank which is being filled, the door may be gradually raised so as to retain the sand in the tank while the water and slime flow over the blocks and thence pass through the proper troughs to the slime-settling-tanks, which are numbered in this mill from 67 to 71. A pipe with a valve connects the box with the trough, so that during the treatment this opening may be closed.

Treatment of the sand by percolation.—While a tank is being charged with sand, as formerly stated, the slime is maintained in suspension by means of wooden hoes, until it overflows, together with the excess water used in grinding, above the blocks placed in the door of the side box, and when all of the blocks have been placed, and the tank is full of sand, the valve of the box is closed, and that of the pipe in the bottom of the tank is opened so that the water held in the sand may drain out through the filter bottom, this drainage being assisted by the application of a vacuum, by means of a vacuum pump attached to this bottom pipe, so that the sand may be dried as much as possible. The surface of the sand in the tank is then leveled by the use of hoes, and crystals of lead acetate are scattered over the surface in the proportion of 20 gm. of lead acetate per ton of sand contained in the tank. A solution of cyanide, containing 0.4% KCN is then introduced below the filter bottom, allowing the solution to enter until it covers the sand in the tank.

The introduction of the first solution from below not only secures a uniform contact of the solution with the whole charge, but also causes an arrangement of the sand grains in a position which is favorable for subsequent leaching. This first solution is allowed to stand on the charge for 3 hours, then the valve in the bottom is opened, and the solution is allowed to drain through the charge for another 3 hours. The surface of the charge is then sprinkled with crystals of mercuric bichloride, in the proportion of 20 gm. per ton of sand, and the second bath of the cyanide solution, 0.4% KCN, is

introduced, this bath being introduced on top of the charge, allowing the stream to fall on a sack so as not to dig up the surface. The second solution is allowed to stand on the charge for 3 hours and then to drain for a further 3 hours. This treatment is followed by four subsequent baths of solution introduced on top of the charge, each bath being followed by a draining for 3 hours, the only difference between these four subsequent treatments and the second, being that neither lead acetate nor mercuric bichloride is introduced in these subsequent treatments.

After the treatment of 36 hours just described the charge is drained and dried by the application of the vacuum pump, and the charge is then turned over in the tank, by laborers working with shovels (this work being contracted for at 5 cents, Mexican currency, per ton), in order to allow the oxygen of the air to come in contact with the sand. The sand is then given six more washes introducing the first wash from below and the others from above, allowing each wash to stand on the charge for 3 hours and to drain for 3 hours, using in the first four washes 0.4% KCN solution and in the last two washes a weak solution containing 0.18% KCN. After this treatment the sand is again dried by the vacuum pump and is then shoveled out of the tank into cars which run on rails underneath the tanks and out of the mill to the dump. The discharge of the tanks costs from 8 to 10 cents, Mexican currency, per ton. The total time of the sand treatment is 86 hours. Each tank treats ordinarily 45 tons of sand. About 155 tons of solution are ordinarily required in the treatment of one tank.

Precipitation.—The strong solutions from the sand treatment, containing 0.4% KCN and 0.136% of protective alkali flow to the precipitation room, where they pass through precipitation-boxes, each of which contains 10 compartments 24 in. by 24 in. by 24 in. filled with zinc-shavings. There are 6 of these boxes and the solution is so divided between them that it flows at the rate of 3½ tons per hour. After the silver and gold have been precipitated from the solution it is pumped to the solution stock-tank where its strength is raised to that required by the addition of strong cyanide solution.

Treatment of the slime by decantation.—The slime and sand finer than 200-mesh resulting from the classification of the sulphide and oxidized ores are collected in 25 settling-compartments, numbers 1 to 25. The compartments 1 to 15 are each 18 ft. by 10 ft. by 5 ft., and are formed by dividing 5 large tanks 18 ft. by 30 ft. by 5 ft. into three compartments each, while the compartments numbers 16 to 25 are each 18 ft. by 15 ft. by 5 ft., and are made by dividing 5 large tanks 18 ft. by 30 ft. by 5 ft. into two compartments each.

First treatment.—The settling-tanks are filled two-thirds full of slime and allowed to settle. The excess water together with any very fine slime which does not settle is then decanted off into large settling-tanks, numbers 41 to 50, for settlement and subsequent treat-

ment. As soon as the decantation is completed the tank is filled with weak cyanide solution, containing 0.18% KCN, 40 gm. of lead acetate are added for each ton of dry slime present, and the slime is stirred by means of compressed air. This air is introduced through an orifice 1/16 in. in diameter in an iron pipe attached to a rubber hose, so arranged that a laborer can move the pipe to all parts of the tank, thus causing a thorough agitation of the slime. The first treatment consists of:

One application of cyanide solution.

2 hours of agitation.

3½ hours of total treatment, including settlement, agitation and transference of charge.

Second treatment.—From the compartments 1 to 25 in which the first treatment is given the slime is transferred to tanks numbers 26 to 40 in which the second treatment is given. These tanks are each 18 ft. by 30 ft. by 4 ft., and in them the slime is treated with 5 solutions each containing 0.18% KCN, agitating during 1½ hours and giving 3 hours of settlement and decantation for each solution. With the first of these solutions 20 gm. of mercuric bichloride per ton of dry slime is added, by placing this salt in a sack of cotton cloth tied to the end of the iron pipe through which the air is entering so that the dissolution of the salt may occur during the agitation. Furthermore, 10 kg. of lime are added with each new solution in order to facilitate the settlement of the slime after each agitation. After the second treatment the slime is transferred to the first washing plant, which will be described hereafter. The second treatment consists of, 5 applications of solution; 7.5 hours of agitation; 22.5 hours of total treatment, including solution, agitation and decantation.

The first and second treatments comprise a total of, 6 applications of solution; 9.5 hours of agitation; 26 hours of total treatment.

Treatment of the very fine slime which did not settle in compartments 1 to 25.—The excess water and unsettled slime from compartments 1 to 25 pass to tanks 41 to 50, each of which is 18 ft. by 30 ft. by 4 ft., for settlement. Thence the excess of water and a small quantity of slime passes to the masonry settling-tanks, where the greater portion of the slime still held in suspension is settled, while the supernatant water is raised by centrifugal and Deane pumps to the tanks above the mill which supply water to the Huntington mills, concentrating-tables and hydraulic-cones. The slime settled in the masonry tanks is at times pumped through a No. 9 Cameron pump to one of the slime treatment tanks for treatment and at times treated with cyanide solution in the masonry tank, according to the exigencies of the work. The slime settled in tanks numbered 41 to 50, and in the masonry tanks, is treated with 7 washes of 0.18% KCN solution, in each of which 1½ hours is occupied in agitation with compressed air, and

3 hours in settlement and decantation. Ten kilograms of lime are added with each wash, and furthermore to the first wash 40 gm. of lead acetate are added per ton of dry slime, and to the second wash 20 gm. of mercuric bichloride are added per ton of dry slime, in the same manner heretofore described. The treatment of the slime in these tanks comprises, 7 applications of solution; 11.8 hours of agitation; 31.6 hours of total treatment.

Treatment of slime in tanks numbers 67 to 71.—The slime collected in these tanks contains also the fine sand which overflows from the sand tanks during the operation of filling these latter. This slime is treated with 6 applications of solution containing 0.18% KCN; 7 hours of total agitation; 22 hours of total treatment; giving the same washes, decantations and applications of chemicals described in the preceding paragraph. All of the slime, after receiving the various treatments heretofore described, is discharged into the first washing-plant for subsequent treatment.

Filtration and precipitation of solutions.—All of the weak solutions, containing 0.18% KCN and 0.146% of protective alkali, resulting from the different washes in the various treatments heretofore described, are decanted from the tanks in which they are used as washes, as soon as they have cleared by the settlement of the slime suspended in them, and are passed to a filter tank.

This tank is 18 ft. by 30 ft. by 4 ft., and has a filter bottom like that described in the sand treatment tanks on top of which is placed a layer of 6 in. of clean sand, with a carpet of jute sacking placed on top of the sand, in order to filter the solutions from the slime suspended in them. The sand layer is changed every five days, as otherwise it would soon fill up with slime which would prevent filtration. From this first filter tank the solution is passed through a second filter tank made in the same manner, and thence passes through the precipitation-boxes filled with zinc-shavings after passing through which it falls into a sump tank from which it is pumped to the stock tank located above the mill, where its strength is raised, to that required, by the addition of strong solution.

First washing plant.—This plant is composed of 16 round wooden tanks, 16 ft. in diameter by 4 ft. deep, and 4 rectangular tanks 15 ft. by 15 ft. by 5 ft. The slime flows to this plant through troughs, and samples are automatically taken as the slime passes through them. The treatment in this plant comprises 2 washes with water, with intermediate decantations, and no cyanide solution is added during this, nor during the subsequent treatment. The slime after this treatment passes through troughs to the second plant.

Second washing plant.—This plant is composed of 20 rectangular tanks, 15 ft. by 15 ft. by 5 ft. Here 2 water washes are given to the slime, with intermediate decantations, and the slime is then discharged into 5 small dams where, as soon as the slime has settled, the supernatant solution is decanted off. The slime is then discharged

by washing out with water into 4 large ponds, where the slime is again allowed to settle and the solution decanted off. In these dams the slime is given a further wash, by stirring up with a hose, and subsequent decantation, if there is time for such operation. From the large ponds the slime is discharged into the river bed.

Solutions from the washing plants.—All of the solutions from the first and second washing plants and from the small and large ponds, flow to 5 masonry filter tanks from which the clear solution passes to 16 precipitation boxes, 3 ft. by 3 ft. by 3 ft., filled with zinc-shavings, and after the metals have been precipitated the solutions are raised by pumps, to be used in subsequent washings. These precipitation-boxes have a capacity of 1 cu. ft. of zinc-shavings for every 1.25 tons of wash water. The solution or wash water from the washing plants, contains from 0.01% to 0.05% of KCN, and carries 6.4 gm. (0.186 oz.) of silver and 0.89 gm. (0.026 oz.) of gold per ton, and it is found that a perfect precipitation of the metals is obtained from these solutions without the addition of any strong solution to the head of the precipitation-boxes. The circulation of the washing solutions is made by centrifugal pumps, as well as Deane and Cameron pumps. About 3,000 tons of washing-solution are pumped every 24 hours. The precipitation-boxes of the washing system are cleared every week and the precipitate mixed with that from the mill.

Precipitation, clean-up, and melting.—In the precipitation department of the mill there are 18 precipitation-boxes, each 2 ft. by 2 ft. by 21 ft., each divided into 10 compartments measuring 24 inches cube. Of these boxes 6 are used for strong solution and 12 for weak solution. The strong solution flows through the boxes at the rate of 3.5 tons per hour, and the weak solution flows at the rate of 6 tons per hour, which in each case represents 2.5 tons of solution for each cubic foot of zinc-shavings. Altogether, counting both strong and weak, 2,700 tons of solution are precipitated every 24 hours, the average content being 10.63 gm. of silver and 2.64 gm. of gold per metric ton. The precipitation-boxes of the strong solution are cleaned every 5 days, while those of the weak solution are cleaned every 7 days.

In the clean-up, after shaking and removing the zinc-shavings, the precipitated slime and short zinc are drawn out of the boxes by a siphon into 6 tanks, where they are treated with dilute sulphuric acid (1 of acid to 7 of water). After the acid treatment the precipitate is washed from 6 to 10 times with water. All of the water from these washings is passed through a Johnston filter-press, from which the clear liquid passes to a sump, to recover any precipitate which might escape through the breaking of a filter cloth, and thence the liquid is discharged into the river bed. From the filter-press the precipitate is placed on two drying stoves. These stoves are simply sheet-iron pans, 6 ft. by 3 ft. by 2 ft., with

round bottoms, placed over a wood fire. The drying takes from 6 to 8 hours, and is usually done at night. The dried precipitate is mixed with the proper amount of flux, composed of soda and borax, and is melted in 4 hours, in graphite crucibles placed in a coke furnace. The slag from the melt is mixed with litharge and a reducing agent and again melted, whereby the greater part of the precious metal contained in the slag is obtained in lead bars.

The bars are sampled by boring above and below with a ¼-in. bit. The bars resulting from the cyanidation of the sulphide ores assay 589 fine in silver and 54 fine in gold, or a total fineness of 643. The remainder of the contents of the bars is mostly lead, due to the addition of lead acetate in the tanks, which is precipitated as metallic lead on the zinc-shavings. The bars resulting from the cyanidation of the oxidized ore usually have a fineness of 506 in silver and 143 in gold, or a total fineness of 649 in silver and gold. The precipitate ordinarily yields 40% of its weight in bars. The slag produced in melting the precipitate from the cyanidation of the sulphide ore assays 36 gm. of silver and 89 gm. of gold per ton. The slag resulting from the melting of the precipitate from the cyanidation of the oxidized ore assays 3,359 gm. of silver and 257 gm. of gold per ton.

Consumption of Chemicals.—Every ton of ore treated in this mill consumes the following chemicals:

	Pounds
Potassium cyanide	2.63
Zinc	1.64
Sulphuric acid	0.45
Lime	39.00

Assays and extractions.—

Sulphide ores:	Silver Grams	Gold Grams	Value
Assay of sulphide ores, gm. per ton	307	30.46	₱45.37
Assay of concentrates (5.58 per cent of weight) gm. per ton	1,596	258	*₱22.02
Assay of tailing from concentrators, gm. per ton.	288	17	₱34.76
Assay of tailing from cyanidation, gm. per ton..	120	1.6	₱6.08
Extraction in cyanidation, per cent	59.35	90.25	79.81%
Total per cent extraction in concentration and cyanidation, per cent	68.57	94.87	86.6%

(*Value extracted in concentrate per ton of original sulphide ore.)

Oxidized ores:			
Assay of oxidized ore, gm. per ton	84.5	14.9	₱23.29
Assay of tailing from cyanidation, gm. per ton..	36.	1.4	₱3.38
Extraction, per cent	57.33	90.64	86.20%

Cost of Treatment.—During the fiscal year 1906-7 the cost per metric ton for grinding, concentrating and cyanidation of sulphide ores was ₱6.35, and for grinding and cyanidation of oxidized ores was ₱5.06.

The New Mill.

The following details of the present practice in the new and modern 700-ton mill of the Esperanza Mining Company were courteously furnished to the author by M. H. Kuryla, then metallurgical engineer for the company, by permission of Charles Hoyle, the general manager of the company.

Ores.—There are two classes of ore treated; sulphide and oxide. The former is composed of quartz, carrying the silver sulphides and the gold with the pyrite, some free gold also being present. A small portion of slate is brought in with the sulphide ore, due to the occurrence of 'horses' in the vein, and an unavoidable amount broken down from the walls of the veins. The oxide ores are mostly quartz of extreme hardness, and a small amount of clayey fine. The daily tonnage consists of 200 tons of sulphide and 400 tons of oxide. The ores are brought in from the mine bins by a 16 in. Robins belt conveyor, separate runs being made on sulphide and oxide. At the point of distribution to the respective mill bins there are two automatic bucket samplers, in series, giving a final sample 0.04% of the total feed. The buckets pass through the entire stream at a uniform rate and at equal intervals, insuring an accurate sample of the mill heads.

Coarse crushing (in 0.12% KCN solution).—The coarse-crushing machinery was originally erected in the old Esperanza mill, and although in service for about eight years, it has been so remodelled and rearranged that very creditable work is performed.

The ore as delivered to the 400-ton sulphide mill bin has already been broken to a ¾ in. size by a 13 by 24 in. Blake crusher—of the Farrell Foundry & Machine Co.'s make—placed at the mine bins. It is fed by Reliance feeders to fifteen 5-ft. Huntington mills, made by the Power & Mining Machinery Co. A 12-ton electric traveling crane serves the entire row of mills. Each mill is driven at 80 r.p.m by a 15-hp. General Electric motor.

Performance of 5-ft. Huntington Mill.

Feed of ore through ¾-in., tons per 24 Hr.	Tons solution per ton ore	Hp.	Discharge through 60-mesh slot screen				Tons 200 per 24 Hr. per Hp.	Tons crushed per		
			%+ 60	%+ 100	%+ 200	%— 200		Die ring	Roller Shell	Set of 4 Screens
15.0	4.0	13.0	1.2	16.1	22.0	60.7	0.70	733	75	75

In case of the oxide ore, a 13 by 24 in. Farrell Blake crusher at the mine breaks the ore to 1½ in. size; and from the 2100-ton oxide-mill bin it is fed by Reliance feeders to a hundred and twenty 900-lb. stamps, erected some ten years ago. The original wooden mortar blocks have been replaced by concrete, 20 stamps per block, and the battery posts are provided with cast-iron shoes bolted to the con-

crete mortar blocks. Another change consisted in bringing all the stamps into one row, the previous arrangement being 60 stamps back to back. As a result of this general overhauling, very creditable work is being done, when both length of service and light weight are considered. A 350-hp. General Electric synchronous motor is directly coupled to the battery line-shaft. The stamps

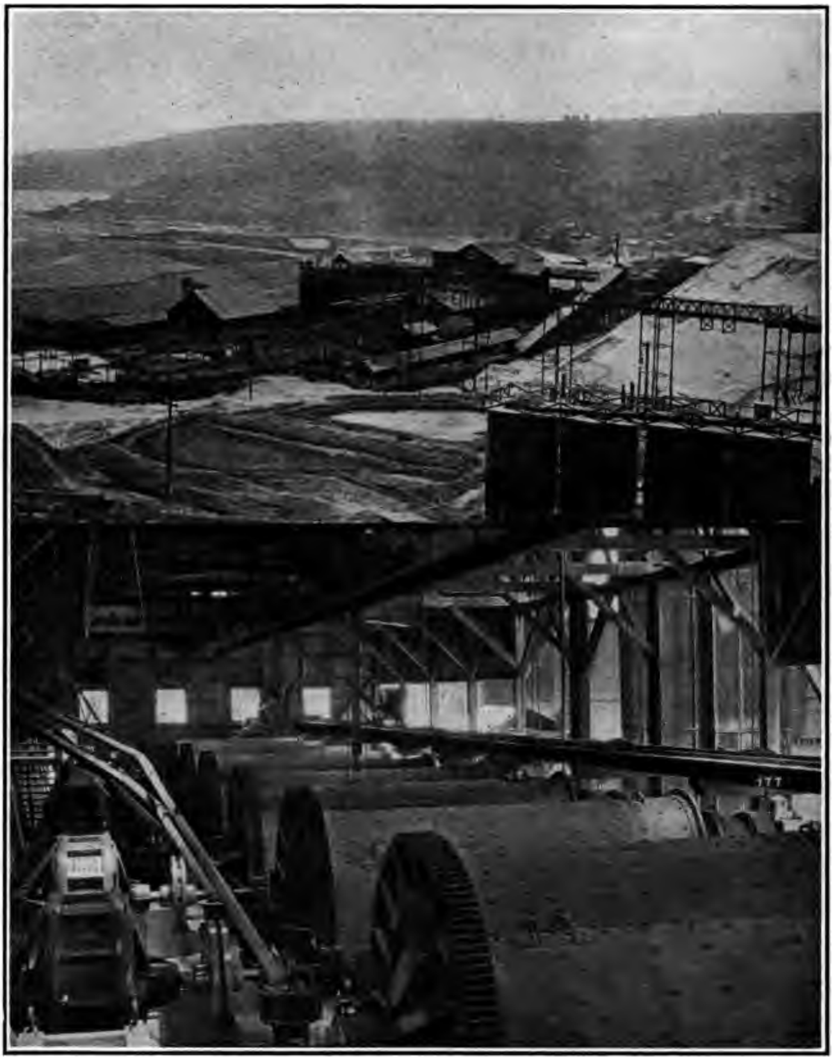


Fig. 8. New Mill of Esperanza Mining Co. General View, Interior of Tube-Mill.

make a hundred and two 6½ in. drops per minute. Six-inch chuck-block and 20-mesh No. 24 steel wire screens are used, giving with 7 tons of solution per ton of ore a duty of 3.4 tons per 24 hours. Tests with 4-mesh No. 16 wire screen, 6 in. chuck, and 6.2 tons of solution per ton of ore, gave a duty of 8.8 tons per 24 hours, with discharge 16% through 200-mesh.

Performance of 900-lb. Stamps.

Feed per stamp of ore through 1½-in., tons per 24 Hr.	Tons solution per ton ore	Hp. per stamp	Discharge through 20-mesh No. 24 wire				Tons 200 per 24 Hr. per Hp.	Tons crushed per		
			% + 60	% + 100	% + 200	% − 200		Shoe	Die	Screen per mtr.
3.4	7.0	2.0	6.8	26.9	19.2	47.1	0.80	167	167	150

Concentration.—The sulphide ore, after being ground through 60-mesh, is carefully classified by three 3-compartment spitzkasten. The spigot sands go to 9 Wilfley tables, where a rough concentration is first made, these concentrates being re-run over two Wilfley tables, where the final shipping grade is made. The overflow of the spitzkasten, consisting of fine flocculent slime, about 95% through 200-mesh, is sent to a storage and thickening tank, which feeds 10 No. 3 Deister slime-tables. These tables make a shipping concentrate at one operation. The concentration serves to remove the rich pyrite, the free gold, and many of the cyanicides, thereby enabling the concentrator tailing to be mixed with the battery oxide pulp and receive identical cyanide treatment. Experiments have been planned to cyanide the concentrate at the plant.

Primary classification.—The concentrator tailing and battery oxide pulp, amounting to a total of about 600 tons of ore and 3600 tons of solution, go to two Esperanza drag classifiers, where the separation is made between coarse sand for regrinding in tube-mills and the fine flocculent slime for agitation in Pachuca tanks. A close separation is demanded, as the sand and slime differ so much in character as to require separate agitation and filter-pressing.

The Esperanza drag classifier consists of a steel box 5 ft. 3 in. wide, with a bottom 15 ft. 7 in. long, sloping 4 in. per foot. Within and at the sides of this box are placed two Ley steel-bushed sprocket chains carried by front and rear sprocket wheels, and to these chains a series of plain wooden strips is attached, 1 in. by 2 in. by 5 ft. long, at 12 in. centres. These strips are carried up the sloping bottom of the classifier at a uniform speed of 15 ft. per minute. The feed is introduced about the central portion of the machine. The sand drops down to and is dragged up by the wooden strips or blades, while the slime overflows into a 2 by 5-ft. collecting-box placed at the desired level of pulp and between the two sprocket

chains. The motion of the drags or strips is uniformly in one direction, and, due to the slow speed, the wear is almost entirely on the wooden strips, which are easily renewed. The power required is almost negligible. The machine readily lends itself to adjustment, and when the character of work required is determined upon, there is practically no variation. An average of two months ½-hr. sampling is submitted:

Performance of one Esperanza Drag Classifier with Average Feed of 275 Dry Tons Ore per 24 Hours.

Product	Sol. Liq. Ratio	Screen analyses							Remarks
		% on 40-mesh	%+ 60	%+ 80	%+ 100	%+ 150	%+ 200	% thru 200	
Feed.....	1: 6.65	1.9	8.5	9.2	1.2	23.9	0.4	54.9	§S:Lwith spray, 1:10
Sand.....	§1: 0.33	9.1	25.2	20.6	2.4	32.4	0.4	9.9	
Slime....	1:12.44	......	......	......	0.9	8.8	0.4	89.9	

Regrinding.—The 300 tons of sand from the primary classifiers carry only 25% moisture; it is diluted to a solid to liquid ratio of 1 to 2.5 to permit of its running to the regrinding plant in a launder of ¾ in. grade. Here it is thickened in sloping bottom dewatering boxes, and mixed with 100 tons per 24 hours of dump sands. The sand dump is made up of a nearly pure quartz sand, 97.8% through 40-mesh, representing the tailing of the leaching tanks of the old Esperanza mill. The regrinding and sliming of this refractory fine quartz sand is particularly difficult when tube-mill tonnage and efficiency must be maintained.

The amount of tube-mill feed, exclusive of the return of the oversize, is 400 tons of sand per 24 hours, fed with 40% moisture to ten No. 5 Krupp seamless tube-mills. These mills are of the following dimensions: inside of shell, 48¾ in. dia. by 19 ft. 8 in. long; inside of liners, 43 in. dia. by 19 ft. 6 in. long. Each mill is driven at 32 r.p.m. by a 60-hp. General Electric motor running at 240 r.p.m. The mill pinion shaft is connected with the motor by a flexible leather-link coupling. The liners are made of cast iron bars connected and reinforced near the bolt holes by ribs. This construction permits of a large percentage of wear, as there is no base or backing to be thrown out. The mills are charged about 4 in. above centre with hard mine quartz, fed in by an automatic double-spiral feed. The tube-mill tailing is elevated by two 18 in. bucket elevators to three Esperanza drag classifiers, and there diluted. These secondary classifiers are of the same design as the primary classifiers, the wooden strips being 1 in. by 3 in. by 5 ft., 24 in. centres, and having a speed of 22 feet per minute. The oversize is returned to the tube-mills. The undersize is sent as 'granular slime' to the Pachuca tanks; first having been diluted to a solid to

liquid ratio of 1 to 5 for purposes of better classification, and then thickened by five 6-ft. and five 8-ft. 65° cones to solid to liquid ratio of 1 to 1½, this thicker pulp being desirable both in the agitation and filter-pressing.

Agitation.—The flocculent slime from the primary classifiers is thickened to a solid to liquid ratio of 1 to 1½ and given 36 hours' agitation in Pachuca tanks. These tanks are six in number, each 14 ft. 10 in. dia. by 44 ft. 8½ in. high, and are worked in the individual system of agitation, i. e., each tank is filled, agitated, and discharged to the mechanical storage-tanks separately. The granular slime from the reground sand is agitated at a solid to liquid ratio of 1 to 1½ in six 14 ft. 10 in. by 44 ft. 8½ in. Pachuca tanks. This agitation was originally of the individual system, and the pulp after agitation was elevated by 18 in. bucket elevators, 54-ft. centers, to the mechanical storage tanks. However, this has been changed to the continuous system. The pulp enters the first tank and by means of a system of internal piping overflows to the other five tanks. From the last tank the pulp overflows near the top to the mechanical storage tanks by gravity, in this way dispensing with the elevators and the more careful attendance demanded by the filling and discharging of each tank individually.

Pulp storage.—The pulp leaving the Pachuca tanks is brought to three storage tanks, where it is kept in agitation and at the head desired for filter-pressing. Each tank is 30 ft. dia. by 12 ft. deep, and equipped with a stirring gear of the Esperanza type. The shaft carrying the stirring arms is enclosed in a 6-in. pipe bolted to the bottom of the tank, which protects the shaft from contact with the pulp. The shaft passes through the tank bottom, and has a ball-and-socket step bearing placed on a concrete pier. The bevel gear and pinion are thereby given a very solid bearing, and the top of the tank has only a steel platform, and no shafting, clutches or gears. The arms make 5 r.p.m., and the three tanks are driven by a 15-hp. General Electric motor.

Filter-pressing.—All of the pulp from the storage tanks is run down to the filter-presses without any previous decanting or washing. The granular slime is treated in three Merrill automatic sluicing filter-presses, each made up of seventy-six 4 in. frames or containers, 3 ft. 8 in. by 5 ft. 8 in. inside dimensions. The sluicing apparatus consists of a 3-in. extra heavy pipe running the full length of the press, and having a 3/16 in. nozzle pointing into each frame. The sluicing pipe is automatically revolved forward and back through an arc of about 180 degrees, by means of the rack, pinion, connecting rod and overhead gearing shown at the head of the press. As the cake is washed away by the action of the jet of water playing upon it, the mixture of slime-residue and water flows into the channel underneath the sluicing pipe and through the outlet cocks to the waste conduit below. Each frame is filled through a continuous

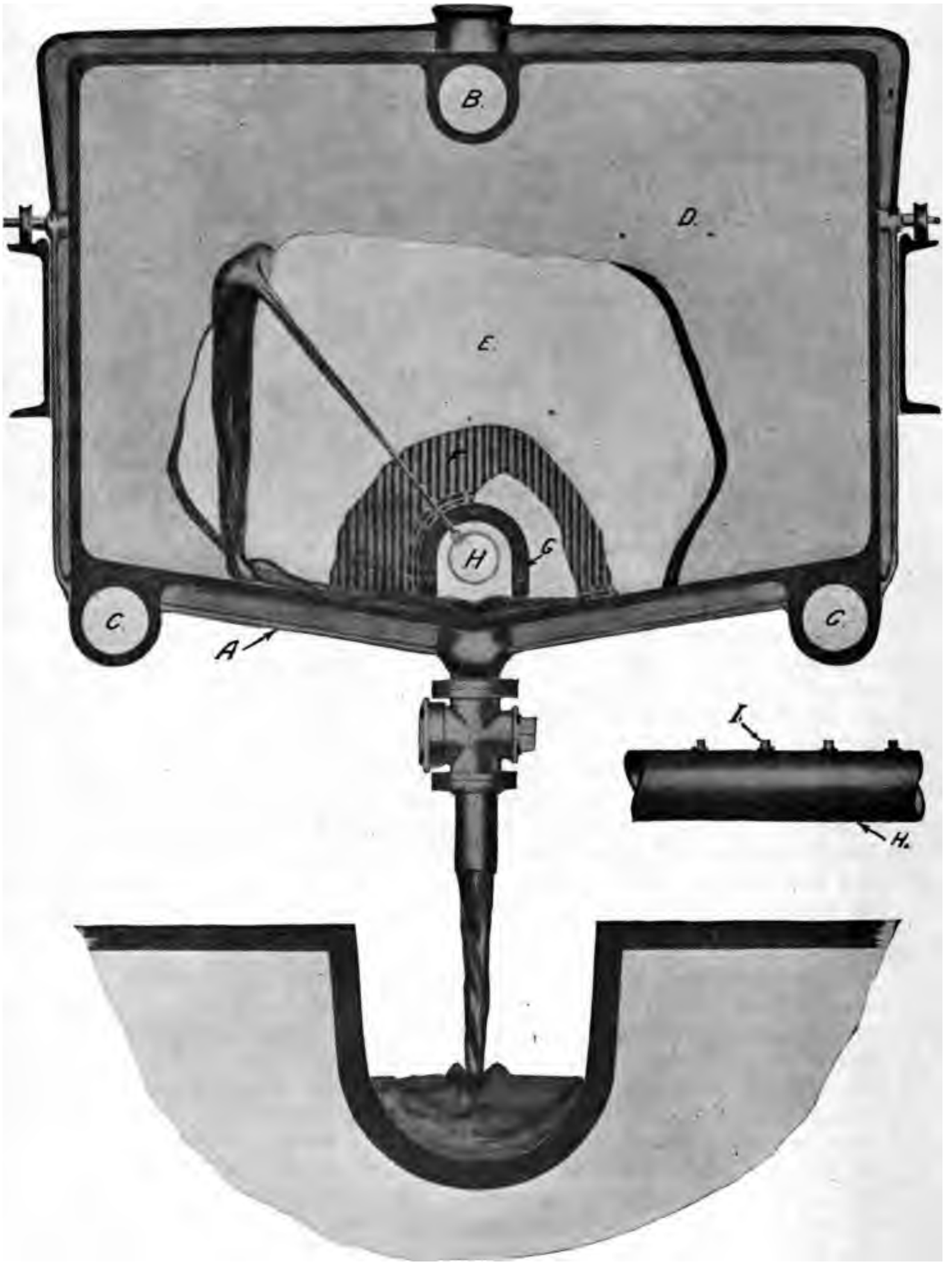


Fig. 9. Merrill Presses as Used at Esperanza Mill.

A, standard container or frame; B, feed channel through which the slime pulp enters each frame; C, channels from which water or solution is drawn off during process of filtration; D, partly sluiced slime cake; E, filter cloth with portion removed showing corrugations of filter plate; F, filter plate; G, horseshoe clamp for holding filter cloth against filter plate· H, sluicing pipe, containing water under 60 to 90 lb. pressure admitted at either or both ends; I, sluicing nozzles at centre of each container or frame.

hollow channel in the top part of the frame, the cake being 4 in. thick and carrying about 20% moisture. The filter-plates between the frames are covered with plain No. 6 canvas, and the wash-water is introduced behind the canvas and is forced through the entire cake in a vertical plane at a uniform rate and with thorough displacement.

The cycle for granular slime is as follows:

	Min.
Filling at 20 lb. pressure	50
Complete washing of 4 in. cakes at 40 lb. pressure	60
Sluicing at 60 lb. pressure	30
Total	140

Capacity per cycle, 21 dry tons of slime.

Capacity per 24 hours at 10 cycles, 210 dry tons of slime, per press.

The flocculent slime is handled in two Merrill automatic sluicing filter-presses, each made up of ninety 3 in. frames, 3 ft. 8 in. by 5 ft. 8 in. inside, and with the same sluicing arrangements found in the above three presses. The method of filling is through the continuous hollow channel in the top part of the frame; but the frame, instead of being filled by a 3 in. solid cake, is filled to such a point that a hollow channel about ¾ in. wide is left down through the entire frame, leaving a 1⅛ in. cake adhering to the No. 6 canvas on each side. The wash-water is then admitted into this hollow space under a pressure of 40 lb. and is forced through both cakes. With this centre-washing this extremely clayey or flocculent slime is capable of being thoroughly washed. Air at a pressure of about 30 lb. is then admitted into the same hollow space between the cakes, and reduces the moisture to about 25%. Previous to turning on the sluicing water, air at a pressure of about 10 lb. is admitted behind the canvas, and causes the cake to peel off into the sluicing stream, leaving the canvas free and clean.

The cycle on flocculent slime is as follows:

	Min.
Filling at 20 lb. pressure	30
Centre-washing at 40 lb. pressure	40
Sluicing at 65 lb. pressure	40
Total	110

Capacity per cycle, 12.5 dry tons of slime.

Capacity per 24 hours at 13 cycles, 160 dry tons of slime, per press.

The sluicing of the five presses requires an average of 4.5 tons of water per ton of slime. The sluiced residue is thickened to 50%

moisture in five 20 ft. by 20 ft. dewatering tanks and discharged into the creek, the recovered water being used for sluicing.

Precipitation.— The filling effluent-solution is called 'weak', and for the first five minutes of each filling is run into the circulating solution circuit. The balance of the weak, or about one ton solution per ton of ore, is precipitated by the Merrill zinc-dust precipitation system. The solution coming from the filter presses is clear enough to precipitate without clarification. The zinc-dust is fed automatically into the suction of the pregnant weak solution pump, and the precipitate recovered in a 22-frame Merrill precipitation-press. The precipitate averages between 60% and 65% of gold and silver bullion. Clean-ups are made at about 6-day intervals. Barrens carry either zero or traces. The pregnant wash, amounting to 2 tons of solution per ton of ore, is precipitated in the same manner as the pregnant weak, only a separate circuit is used, and two 17-frame Merrill precipitation-presses. On account of much lower bullion value of heads, the precipitate averages about 55% of bullion. The barrens are kept at zero or traces. Ten-day clean-ups are used.

The precipitate is air-dried in the presses. Melting is done in two No. 60 Steele-Harvey tilting furnaces, oil-fired. A steel dust-chamber connects with hoods placed over the furnaces.

Consumptions per wet metric ton:

	Kgm.
Sodium cyanide (129% KCN)	0.510
Lime	3.500
Zinc-dust	0.195
Lead acetate	0.050

CHAPTER VII.

CYANIDE PRACTICE OF THE GUANAJUATO CONSOLIDATED MINING & MILLING CO.,

Guanajuato, Mexico.

The following description is taken in part from the article entitled 'Cyanidation of Silver Ores at Guanajuato', by Bernard MacDonald, published in the *Engineering and Mining Journal* of April 4, 1908, and in part from notes taken by the author during a visit to this mill in March, 1909. The ores treated in this mill contain 85% insoluble, 6% iron, 2.5% sulphur and 2% lime, and assay on an average 450 gm. of silver and 2.4 gm. of gold per metric ton. The silver occurs as sulphide and sulphantimonides, and the gold is found associated with the silver sulphide.

Preliminary Treatment.

Crushing and milling.—The ore coming from the mines is crushed in 2 Blake crushers to pass through a 2-in. ring, sorted to separate the waste, and dumped into the ore-bins behind the batteries where it is mixed with 10 kg. of lime per ton of ore in order to neutralize the acidity of the ore due to the decomposition of the pyrite. From this bin the ore is fed to the batteries by Challenge feeders. There are 80 stamps in this mill which weigh 1050 lb. each, drop 7½ in. and have 104 drops per minute. Seven and two-tenths tons of previously precipitated 0.025% KCN solution, per ton of ore, are admitted to the batteries, and the stamps have a duty of 3.3 tons of ore per stamp or a total crushing capacity of 264 tons of ore per day through a No. 10 Harrington and King slot metal screen, which is equivalent to a 50-mesh screen. A sizing test of the battery pulp shows the following composition:

Screen	+40	+50	+60	+80	+100	+120	+150	+200	−200
Per cent.	0.89	2.71	6.92	10.71	0.34	10.41	7.65	1.72	58.65

Dewatering and concentration.—The battery pulp contains too much water to yield good results in concentration, and consequently the pulp from each battery of five stamps is run into a two-compartment spitzkasten for the purpose of dewatering the pulp before concentration. Each compartment of the spitzkasten measures 22 in. cube, and the discharge of thickened pulp in the vertex is so regulated that 12% of solution contained in the pulp, carrying a

small quantity of fine slime, is allowed to overflow from the second compartment whence it flows to the slime settling tanks, while the thickened pulp, containing the proper amount of water, falls from the vertices of the two compartments upon Wilfley concentrators. There are 25 concentrators in this mill, of which 12 are Wilfleys, 4 are Standards, and 13 are Frue vanners. The concentrates from all of these tables are conducted by pipes to storage tanks in the concentrate room, and usually contain 2½% of the weight and 50% of the value of the original ore. These concentrates, containing 42% of iron and 6% of silica, are sold to the ore buyers. The fine slime and excess of water which discharge over the last five feet of the tail of the Wilfley and Standard concentrators are cut off and conducted to the slime-collecting tanks while the balance of the tailing from these tables is fed to the Frue vanners for further concentration.

Classification into sand and slime.—The tailing from the Frue vanners, which contains all of the battery pulp, with the exception of the concentrate and that portion of the slime already separated, still contains a considerable portion of slime and as a different treatment is given to the sand and slime, a separation of the two products is necessary. This separation is effected in 10 hydraulic-cone classifiers of sheet steel, 3½ ft. in diameter by 3½ ft. deep. The sand, together with a small quantity of slime, settles in the bottom or vertex of the cones, whence it is discharged into troughs which carry it to the sand-elevators, while the water entering at the vertex of the cone rises and overflows at the top, carrying with it the slime which is conducted through troughs to the slime-settling tanks already mentioned. The sand-elevators consist of two belt elevators, the centres of whose pulleys are 65 ft. apart, whereby the sand is elevated to the troughs leading to the sand-collecting tanks.

The sand, separated by the hydraulic cones just described, still contains a considerable quantity of slime, for whose elimination it is customary to allow the sand to discharge from the trough above mentioned into cone classifiers, 18 in. diameter by 18 in. deep, placed above each sand-collecting tank. The slime which overflows from these cones is conducted to the slime-collecting tank, while the sand discharged from the vertices falls on one side of the sand-collecting tanks, where it piles up with a natural slope towards a gate on the other side of the tank. This gate is formed of perforated strips of 2-in. boards, covered by a strip of canvas which permits the slime and excess solution to overflow and unite with the overflow from the cones above the tanks. The canvas is arranged in a roll which may be raised as more wooden strips are added to the gate.

In order to assist in the separation of the sand and slime a boy stands over the tank with a hose and directs a small stream of water upon the surface of the sand as it is being deposited, so that any slime still remaining in the sand is washed out, and the sand

is left in perfect condition for subsequent treatment by percolation. There are seven sand-collecting tanks, shown in Fig. 10 in the background, above the tracks. These are 26 ft. in diameter by 5 ft. deep, arranged with filter bottoms, drainage pipes, and doors in the bottom for the discharge of their contents. The classification of the pulp divides it into 47% by weight of sand and 53% of slime.

Sand Treatment.

First treatment.—It usually takes 15 hours to fill a sand-collecting tank, at the end of which time it will contain 90 tons, dry weight. The sand is allowed to drain for 15 hours, thus reducing its moisture contents to 18%, and a sample is then taken for assay by pushing a hollow tube through the sand from top to bottom, and from this the assay value of the tank is determined. Twenty tons of 0.5% KCN solution and 12 kg. of lead acetate are then added and allowed to saturate during 12 hours. The stream of solution entering the tank falls into a basket containing 25 kg. of dry sodium cyanide, and dissolves it fast enough to keep the strength of the solution, when mixed with the water contained in the sand, between 0.5% and 0.52% KCN. After the 12-hour saturation, just mentioned, during which the silver and gold are being dissolved, the valves located below the filter-bottom are opened and the solution is allowed to percolate through the sand, which requires about three hours, and the sand is then allowed to drain and aerate for a further three hours.

The operation of saturation and percolation is then repeated, under the same conditions of solution, salts added and time of contact, except that in this second operation the sand is allowed to drain and aerate during 12 hours, thus reducing its moisture contents to 14%. The total time of treatment in the collecting-tanks is 78 hours. The solutions resulting from the percolations and drainings just mentioned run to the precipitation-boxes for strong solution, where the silver and gold contents are precipitated on zinc-shavings. The pipes carrying these solutions have pet-cocks in the bottom for taking samples from which the total active cyanide, protective alkali and silver and gold contents are determined and thence the results of the treatment at any stage. It is found that from 35% to 40% of the silver and gold contained in the original ore is extracted in the collecting-tanks.

Transfer of charge.—As soon as the treatment just described is concluded the four doors in the bottom of the collecting tanks are opened and the sand is discharged into double side-discharge cars running on rails which pass beneath these tanks and above the tanks in which the sand is given its second treatment. A handful of sand taken from each car and thrown into a box whose interior capacity is two cubic feet, serves as a sample. This sample is weighed to determine its specific gravity and thence the weight

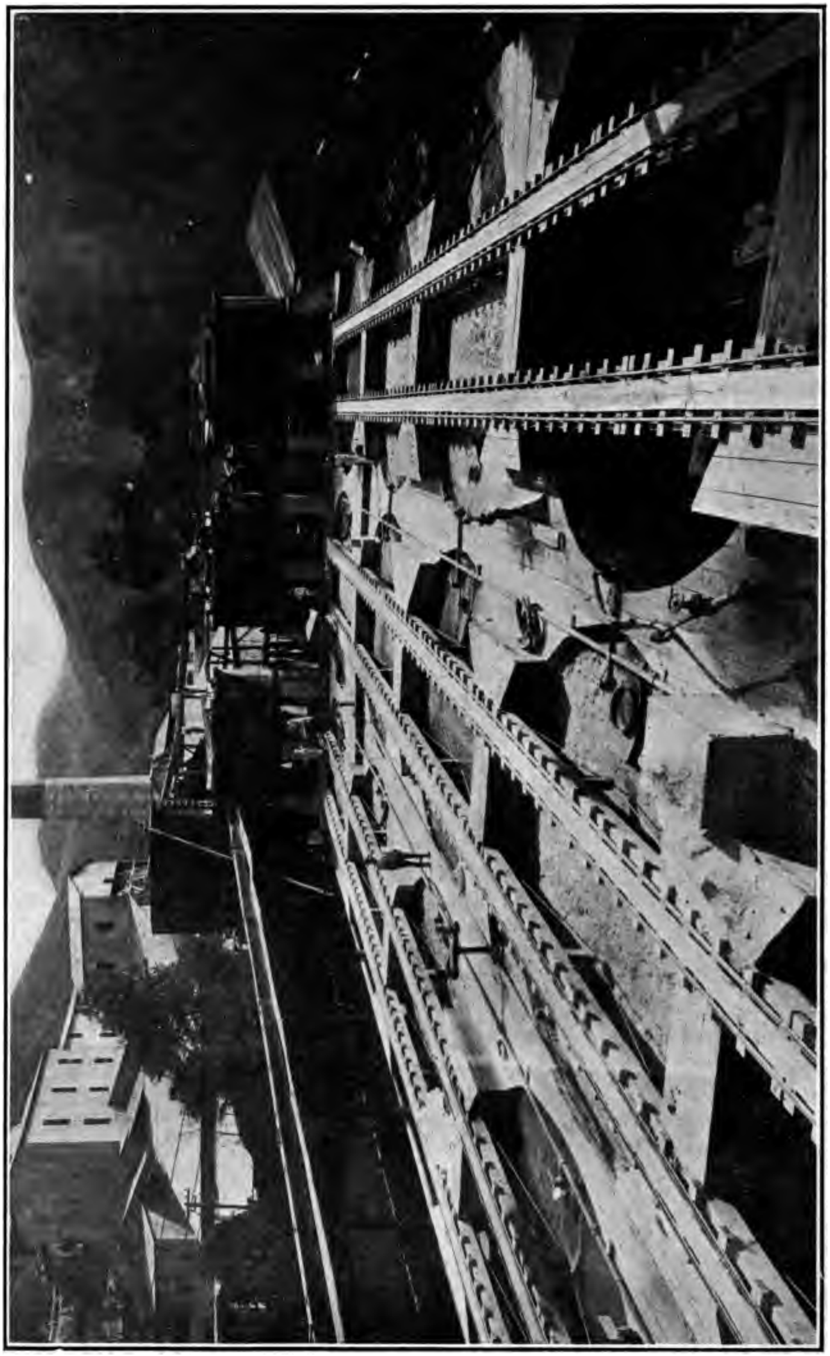


Fig. 10. Sand Plant of Guanajuato Cons. M. & M. Co.

of the charge introduced into the second-treatment tanks; it is also assayed for moisture (ordinarily 12% to 14%) and for its silver and gold content in order to determine the extraction during the first treatment. As soon as the sand is all discharged the doors are closed and the tanks are ready to receive another charge of sand.

Second treatment.—The tanks in which the second treatment of the sand is given, shown below the tracks in Fig. 10, are 15 in number, each 26 ft. in diameter by 6 ft. deep, arranged with filter bottoms, drainage pipes and two discharge doors in the bottom. The treatment in these tanks is continued during 11 days. While charging these tanks, which ordinarily takes 6 hours, 450 kg. of lime is scattered through the charge. Thirty tons of 0.5% KCN solution with an additional 25 kg. of dry cyanide is then added and allowed to saturate for 24 hours, the valve below the filter-bottom is then opened and the solution is allowed to percolate through the charge for 6 hours, running thereafter directly to the precipitation-boxes for strong solution. Thirty tons more of 0.5% KCN solution is then added and then, during 6½ days the solution is allowed to percolate through the charge, adding new solution in a continuous stream on top as fast as it seeps through the charge.

At the end of this time, the charge is treated by percolation for two days with 0.2% KCN solution, and afterwards for 18 hours with water to wash out the last traces of valuable solution. A sample is taken of the sand and it is then discharged by a stream of water through a hose which washes out the tank, discharging its treated sand into the river. The operations of sampling and discharge require 6 hours, so that the total time of the sand-treatment, including first treatment, transference of charge and second treatment, is 14 days. During this treatment 12 kg. of lead acetate are added to the tank, or 117 gm. per ton. A regular set of books is carried for keeping the accounts of each tank treated, in which are noted the time of treatment, methods employed, amount of chemicals used, results obtained at the end of each stage of the treatment and the percentage extraction at the end of each day of the treatment.

Slime Treatment.

Slime-collecting tanks.—The slime-collecting tanks, consist of two parallel rows of masonry tanks, 12 in each row, measuring 9 ft. by 9 ft. by 8 ft. deep, which are located below the concentrator floor, and furthermore there are 3 spitzkasten, each 8 ft by 8 ft. by 30 ft. long, each of which is divided into five compartments. The stream of slime, which has been separated from the sand as already explained, is passed through one row of tanks until that row is filled with partly-dewatered slime, when the stream of slime is allowed to run through the second row of tanks, while the slime in the first row is allowed to settle. The clear water is decanted off from on top of the slime in each tank and flows to sump-tanks,

whence it is pumped to tanks above the mill which supply the batteries with water. The settled slime, containing 75% of water, or 3 of water to 1 of dry slime, is drawn off from the tanks through a gate in the bottom, and runs through troughs to the agitation-tanks. The gates are then closed, and the current of slime is again turned into this series of tanks while the other series is being settled, decanted and discharged in the same manner.

Cyanidation of the slime.—There are 14 agitation-tanks, each 30 ft. in diameter by 10 ft. deep (shown in Fig. 11) arranged with mechanical agitators and the piping necessary to decant the supernatant solution and to transfer the slime from one tank to another by pumps. The slime from the collecting-tanks is allowed to enter

Fig. 11. Slime Plant of Guanajuato Cons. M. & M. Co.

one of the agitation-tanks until it is filled to a depth of 8 ft., which is equivalent to a charge of from 35 to 40 tons of dry slime. As it enters the agitation-tank the stream of slime falls into a basket containing a sufficient amount of dry sodium cyanide to raise the strength of the solution contained in the slime to 0.2% KCN. The dry cyanide in the basket dissolves slowly but completely. The remaining 2 feet of tank space, after filling with slime to the depth of 8 ft., is filled with 0.2% KCN solution. This addition gives the slime in the agitation-tank approximately 85% of solution, or a dilution of 5 of solution to 1 of dry slime, which is that which gives the best results in affording facility in settling and subsequent decantation. The agitation of the slime is commenced as soon as the slime enters the tank, both the mechanical agitation and the

agitation by centrifugal pumps which draw the slime from the bottom of the tank and discharge it on top of the charge in the same tank, and continues without interruption for 24 hours.

As soon as the tank has received its charge of slime 8 kg. of lead acetate and 5 cans of sifted burned lime (containing a total of 125 lb. CaO) are emptied into it. The lime not only serves to neutralize the acidity but also to coagulate the slime and cause its rapid settlement. After 24 hours the agitation is suspended and the charge is allowed to settle for 6 hours, at the end of which period it is found that there is about 3 ft. of clear solution on top of the settled slime. This solution is decanted off and flows to the clarifying-tanks. The agitation is again begun, filling up the tank with more 0.2% KCN solution, and adding the same quantities of lead acetate and lime as before, and continued for half an hour, or until the solution just added is thoroughly mixed with the charge. The charge is then allowed to settle for half an hour, and 18 inches of clear solution are decanted off. This operation of adding new solution, and lime, with its agitation, settlement and decantation is repeated six times.

After the eight treatments just described the charge is given two water washes, after each of which the charge is agitated, allowed to settle and the supernatant solutions decanted and conducted to the same clarifying tanks already mentioned. The charge is then transferred by pumps to one of the two deep settling tanks where it is mixed with weak solution which has previously been pumped to this tank filling it to a depth of 6 feet. The charge is allowed to settle in this tank as long as possible, which time depends on the amount of ore treated, etc. The clear supernatant solution is decanted and conducted to the clarifying-tanks, while the slime which has settled is pumped to the Burt filters, whence the filtered solutions pass to the precipitation-boxes, while the filtered slime is discharged into the river.

Final Treatment and Cost.

Assay of slime and solution.—The slime and solution are assayed at each stage of the treatment in order to determine the quantity of silver and gold still present in the slime as well as the percentage of extraction obtained at the end of each operation and the percentage of cyanide in solution. The dry weight and assay of the slime entering the agitating-tanks is determined from samples taken at frequent intervals across the current of slime falling into the tank. Of this sample one cubic foot is taken from which a sample is taken to determine the percentage of moisture and the assay of the slime after drying, from which is calculated the weight and value of dry slime contained in each tank. The slimes discharged from the deep settling-tanks and from the Burt filters are sampled and assayed in the same manner, determining the cyanide content

as well as other constituents, and the total extraction is calculated from the difference in assays.

Clarifying solutions.—All of the solutions decanted off the slime are clarified before precipitation of their metallic contents. The decanted solutions first run to an intermediate sump-tank, whence they are pumped to the first of the clarifying-tanks, each of which is 26 ft. in diameter by 12 ft. deep. The solution flows from the first to the second clarifying-tank, passing through two or three curtain filters made of jute or cocoa matting. From the second tank the filtered solutions flow to the precipitation-boxes.

Precipitation.—The solutions enter the precipitation-room through two pipes, one carrying strong and the other weak solution. The quantity of solution precipitated is calculated by measurement in the tank in which the solution is stored before precipitation, and the value of the solution is calculated by samples taken before and after precipitation. This sample is taken through a nipple and ¼ in. pet cock set in the bottom of the pipe leading the solution to the precipitation-boxes, opening the cock so that the solution falls drop by drop into a 2-litre bottle at such a speed that the bottle will contain a sample representing 24 hours flow. This sample is assayed for silver, gold, active cyanide and protective alkali. From the quantity of solution and the assay value before and after precipitation the values precipitated are calculated.

There are 14 zinc-boxes of four compartments each. These compartments are 4 ft. long by 3 ft. wide by 2 ft. deep, having double partitions in the end which cut off 4 in. of the length so that each compartment has a capacity for 20 cu. ft. of zinc-shavings, and for filling the 56 compartments with zinc-shavings requires four tons of zinc. The consumption of zinc is 0.814 kg. per ton of ore treated or 0.05 kg. per ton of solution in circulation.

The solutions, after passing through the zinc-boxes, flow to sump-tanks, from which they are pumped to solution-storage tanks, where by the addition of dry cyanide their strength is raised to that required in the treatment. The amount of solutions used in this mill per ton of ore treated is as follows: In the sand treatment:—7 tons of 0.5% KCN solution; 1.4 tons of 0.2% KCN solution; and 0.7 tons of wash water. In the slime treatment:—6.4 tons of 0.2% KCN solution. This makes a total of 15.5 tons of solution per ton of ore treated, or for the 260 tons of ore milled daily 4,030 tons of solution are precipitated, using 1 kg. of zinc-shavings per ton of solution to precipitate the silver and gold contents.

Clean-up and melting.—The zinc-boxes are cleaned up three times a month in the following manner: The precipitated silver and gold, deposited on the zinc-shavings are shaken off by agitating the shavings in the same compartment, and fall to the bottom, whence, upon opening a valve in the bottom of the compartment, they fall into a trough which carries them to a sump-tank, into

which they fall through a 60-mesh screen which retains the short pieces of zinc. These zinc-shorts, after being scrubbed with a broom on the screen, are removed and placed in layers 2 in. thick, alternating with layers of long zinc-shavings, in the precipitation-boxes.

The precipitated slime which passes through the screen and falls into the sump-tank is pumped to a Johnson filter-press, from which the pressed cakes of slime are removed and placed in a large bowl, in which they are weighed and sampled to determine their dry weight, and are then mixed, still moist, with a flux composed of 7% of the dry weight of the slime of sodium bicarbonate, 15% of borax-glass and from 3 to 5% of quartz sand. The filter-cakes usually contain 30% of moisture, 52% of silver, 3% of gold and varying quantities of zinc, copper, lead, sulphur, and silica. After mixing the moist precipitate with the flux, the mixture is placed in shallow pans which are then placed in a stove, heated by steam-coils, for drying. As soon as dry the mixture is emptied from the pans into No. 300 graphite crucibles in a coke furnace, arranged with pipes introducing compressed air below the grate bars to increase the heat, and as soon as melted the contents of the crucibles are poured into molds, obtaining thereby bars which weigh 32 kg. and assay 860 fine in silver and 5 fine in gold, the other constituents of the bars being zinc and lead. The bars are sampled by borings ¾ in. deep on top and bottom, and upon the assays of these samples the bars are sold to the ore buyers. The consumption of chemicals and cost of the precipitation and melting of one kg. of fine silver and gold, taking the average cost during a period of six months was as follows:

Zinc, 4,830 kg	₱1.680
Cutting the zinc into shavings	0.096
Borax, 370 kg	0.222
Sodium bicarbonate, 224 kg	0.049
Crucibles	0.223
Total per kilogram of pure silver and gold	₱2.27

Extraction.—The total extraction obtained in the treatment of concentration and cyanidation previously described is 87% of the silver and gold contained in the original ore.

Chemical consumption.—The average consumption of chemicals during a year, per ton of ore treated, was as follows:

	Kilograms
Sodium cyanide	1.698
Lead acetate	0.177
Burnt lime	46.3
Zinc	0.814

(Note:—The large consumption of lime was due to the fact that the mine water used in the treatment contained a large quantity of acid.)

Cost.—The cost of treating the ore of this company, aside from mining, per ton of ore is as follows:

Crushing and sorting.

Labor	₱0.0873	
Electric power	0.0253	
Material and repairs	0.0364	
Total cost of crushing and sorting		₱0.1490

Hauling ore from mine to mill.

Labor	₱0.0068	
Electric power	0.0050	
Material and repairs	0.0040	
Total cost of hauling		₱0.0158
Total cost of crushing, sorting and hauling		₱0.1648

Milling and concentration.

Labor, including mechanics, blacksmiths and carpenters	₱0.26	
Materials for mechanics, blacksmiths and carpenters	0.20	
Electric power	0.52	
Assay office	0.03	
General office expenses	0.03	
Expenses of mill	0.05	
Fire insurance	0.01	
Total cost of milling and concentration		₱1.10

Cyanidation.

Labor	₱0.41	
Material, including cyanide	2.06	
Electric power	0.26	
Assay office	0.12	
General office expenses	0.14	
Expenses of mill, salaries, etc	0.24	
		₱3.23
Total cost of crushing, sorting, hauling, milling, concentrating and cyaniding		₱4.4948

(Note—The electric power costs ₱160 per Hp. per year.)

Possibility of Cyaniding Low-Grade Ores.

An examination of the preceding data demonstrates that under favorable conditions, namely, having a large deposit of ore which can be easily mined in large quantities and at a cost not exceeding ₱5 per ton, having cheap power and a mill whose daily capacity is at least 260 tons, and having an ore easily treated by cyanidation, it is possible to mine, mill, concentrate and cyanide an ore containing as low as 0.400 kg. of silver per ton at a profit, as may be seen in the following estimate:

Ore containing 400 gm. at 3.7 cv. per gm. gives a gross value of ₱14.80 per ton	₱14.80	
Cost of mining		₱5.00
Cost of crushing, hauling, milling, concentrating and cyaniding		4.50
Loss in treatment		1.92
Taxes and expenses in selling concentrate and bullion, 7 per cent		0.90
Total cost		₱12.32
Profit		2.48
	₱14.80	₱14.80

CHAPTER VIII.

CYANIDE PRACTICE OF THE REAL DEL MONTE Y PACHUCA CO.,

Pachuca, Hidalgo, Mexico.

The following details as to the practice in the mills of this company were courteously furnished to the author by V. B. Sherrod, superintendent of cyanidation, by permission of the general manager, E. P. Merril. This company has two mills, called, respectively, Loreto and Guerrero, and as the practice in them differs, they will be described separately.

Loreto Mill.

This mill, shown in Fig. 12, is in the city of Pachuca, and consists of an old mill formerly used for the treatment of ore by the patio process, which has been remodeled and arranged with the necessary machinery required for cyanidation, for a capacity of 300 metric tons of ore per day.

Crushing the ore.—The ore coming from the mines falls upon a grizzly with 1½ in. openings. That which passes through the grizzly falls into the ore-bin which supplies the Chilean mills, while the larger particles which do not pass through the grizzly are crushed in 4 Blake crushers, 7 by 10 in., with jaws set to 1½ in. opening, and then fall into the bin which supplies the stamp batteries.

Sampling.—The tenth part of the ore which passes through the grizzlies and crushers is cut out by mechanical means and passes through a set of rolls, 12 by 20 in. Then 10% of this product is cut out and passed through a set of rolls 12 by 14 in., and finally 5% of this last product is mechanically cut out for the final sample of the ore which is taken to be assayed.

Chilean Mills.—There are 14 Chilean mills giving 14 revolutions per minute, each mill having two wheels 2 metres in diameter the iron tires of which are 5 in. thick. These mills each grind 22 tons of ore per day through a 40-mesh screen using from 8 to 10 tons of 0.26% KCN solution per ton of ore ground. A screen sizing-test of the pulp from these mills shows the following composition:

Screen...........	+40	+60	+80	+100	+120	+150	+200	—200
Per cent.........	0.25	2.5	7.5	10.25	2.5	10.5	8.25	54.0

Each of these mills requires 15 hp., or 0.7 hp. per ton of ore.

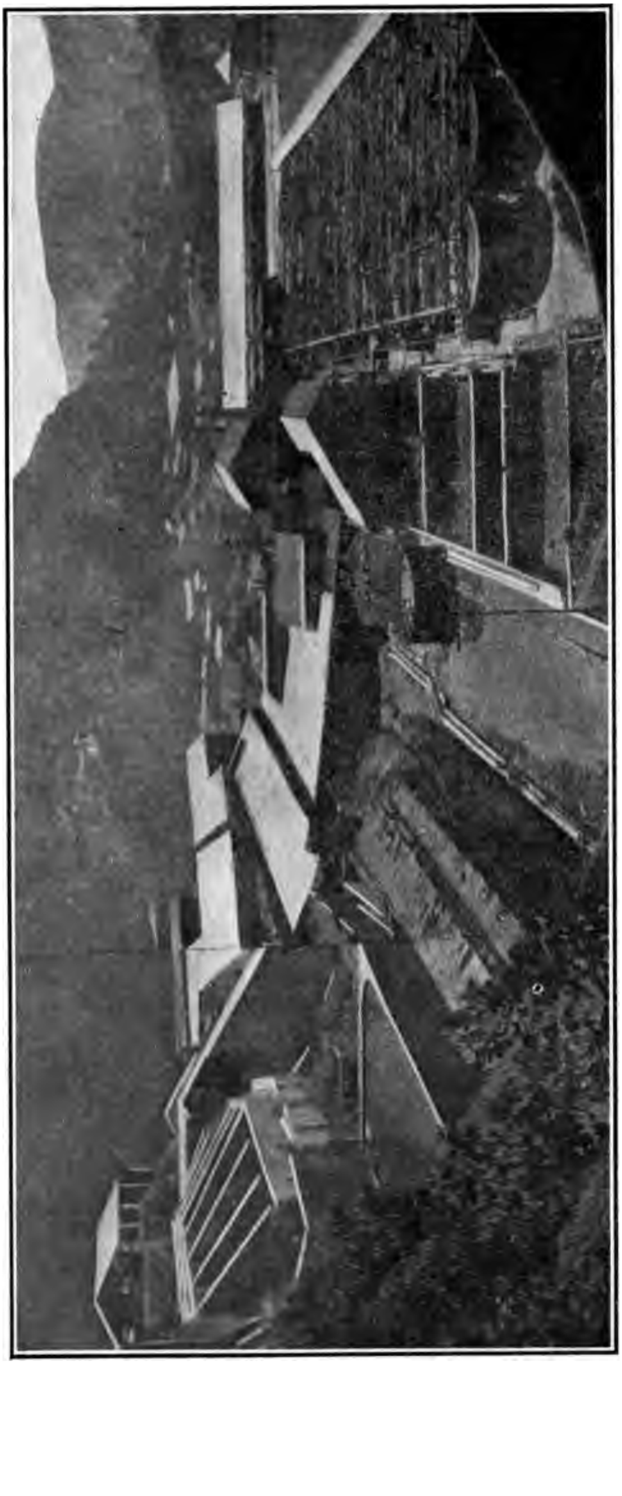


Fig. 12. Loreto Mill of the Real del Monte y Pachuca Co.

Stamp Batteries.—There are 40 stamps in this mill also, each weighing 1,050 lb., 7-in. drop, 106 drops per minute, 40-mesh screens set 2 ½ in. above the dies, and each stamp grinds 2½ tons of ore per 24 hours, using from 8 to 10 tons of 0.26% KCN solution per ton of ore. A screen sizing-test of the battery pulp shows the following composition:

Screen...........	+40	+60	+80	+100	+120	+150	+200	—200
Per cent.........	0.4	15.1	11.7	9.9	1.6	6.5	6.0	45.5

In order to grind 95 tons of ore in the batteries requires 98 hp., or 0.97 hp. per ton of ore. Comparing the results of the grinding in Chilean mills and stamp batteries, both as to fineness and horsepower required, shows that the former grind finer with less power.

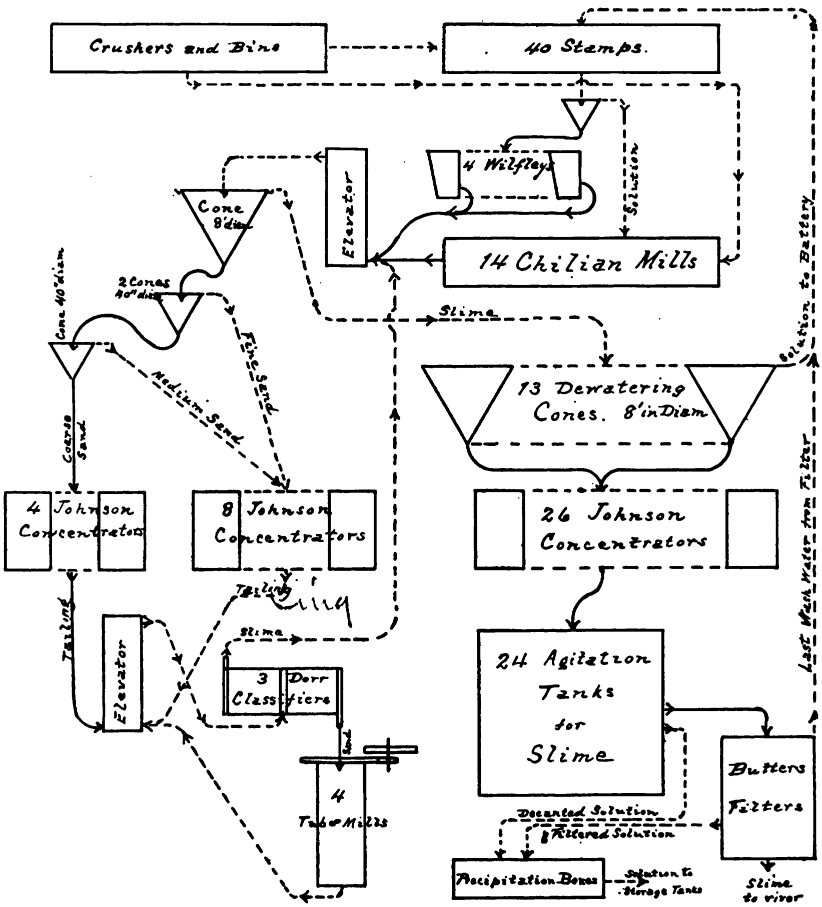


Fig. 13. Flow Sheet of the Loreto Mill.

It has also been determined that the wear on the grinding parts is about equal, but that the mills are more expensive to repair and renew wearing parts. The flow sheet, shown in Fig. 13, graphically represents the flow of the pulp through the mill from the crushers to the agitation tanks.

Classification and first concentration.—From the stamp batteries the pulp passes to a dewatering cone, in order to secure a less watery pulp for concentration on the Wilfleys. The slime which overflows from this cone subjected to a screen sizing-test shows the following composition:

Screen	+60	+80	+100	+120	+150	+200	—200
Per cent	0.0	0.25	1.1	0.5	2.65	5.5	87.35

The ore discharged from the vertex of this cone is distributed among 4 Wilfley tables and a screen sizing-test of this pulp, together with the assay of each separate screen product showing its silver contents in kilograms per ton gave the following results:

Screen	+40	+60	+80	+100	+120	+150	+200	—200
Weight, per ct.	0	22.0	17.8	16.8	2.4	10.44	7.6	20.2
Assay		0.530	0.668	0.770	0.764	1.058	1.355	1.590

A screen sizing-test of the tailing from the Wilfley tables, together with the assay of each separate screen product showing its silver contents in kilograms of silver per ton, gave the following results:

Screen	+40	+60	+80	+100	+120	+150	+200	—200
Weight, per ct.	0	25.9	19.36	15.2	2.96	9.3	6.08	19.4
Assay		0.426	0.432	0.424	0.410	0.460	0.458	0.576

Classification, regrinding, and final concentration.—The tailing from the Wilfleys runs to the foot of a belt-elevator where it is joined by the pulp from the Chilean mills, and both together are elevated and discharged into a cone classifier 8 ft. in diameter. A screen sizing-test on the slime which overflows from this cone gave the following results:

Screen	+60	+80	+100	+120	+150	+200	—200
Per cent	0	0.1	1.0	0.6	2.2	4.6	89.1

This slime flows to 13 dewatering cones 8 ft. in diameter, whence the dewatered slime flows to 26 Johnston concentrators. The sand discharged from the vertex of the 8 ft. cone classifier, upon having a screen sizing-test made of it, showed the following composition:

Screen	+40	+60	+80	+100	+120	+150	+200	+200
Per cent	0.25	2.75	8.6	17.0	2.1	15.6	11.8	39.35

This sand flows to 3 cone classifiers 40 in. in diameter, whence the fine sand which overflows passes to 8 Johnston concentrators while the sand discharged from the vertex flows to another cone classifier 40 in. in diameter from which the fine sand which overflows passes to the same 8 Johnston concentrators, while the coarser sand discharged from the vertex flows to 4 other Johnston concentrators. All of the cones previously mentioned are gravity cones, no water being introduced into the bottom of them for sizing purposes.

A screen sizing-test of the sand discharged from the vertex of the last mentioned cone, which represents the heads of the 4 Johnston concentrators, together with the assay of each separate screen product showing its silver contents in kilograms per metric ton, gave the following results:

Screen.........	+40	+60	+80	+100	+120	+150	+200	—200
Weight, per cent	0	3.4	8.9	17.4	5.36	15.1	11.64	35.7
Assay.........		0.446	0.428	0.476	0.532	0.653	0.836	1.450

A screen sizing-test of the tailing from these concentrators, together with the assay of each separate screen product, showing its silver content in kilograms per metric ton, gave the following result:

Screen.........	+40	+60	+80	+100	+120	+150	+200	—204
Weight, per cent	0	3.2	8.88	17.34	4.6	15.74	11.98	36.00
Assay.........		0.450	0.432	0.432	0.420	0.429	0.413	0.520

All of the tailings from the Johnston concentrators which receive the fine and coarse sands, flow to the foot of a belt-elevator by which they, together with the reground product of the tube-mills, are elevated and discharged into 3 Dorr classifiers.

A screen sizing-test of the tailing entering the Dorr classifiers, and of the slime and sand discharge of the same classifiers, gave the following results:

Screen...........	+40	+60	+80	+100	+120	+150	+200	—200
Tailing feed, per ct	0.2	4.0	7.9	11.5	3.5	16.5	14.2	40.2
Slime overflow "	0.0	0.0	0.05	0.3	0.1	5.75	7.3	85.0
Sand discharge "	0.2	5.3	9.5	17.7	5.5	20.2	17.8	22.3

The slime overflow from the Dorr classifiers flows to the 13 dewatering cones already mentioned, while the sand is discharged into 4 tube-mills for regrinding. These tube-mills are numbered from 1 to 4, No. 1 being a Krupp mill 4 ft. in diameter by 18 ft. long, Nos. 2 and 3 being Abbé mills, 4½ ft. in diameter by 15 ft. long, and No. 4 a Denver Engineering Works mill 5 ft. in diameter by 20 ft. long. A careful series of tests was made to determine the relative efficiency of these tube-mills, under exactly similar conditions, with

the result that it was found that 1 hp. would regrind so as to pass through 200-mesh 0.57 kg. of tailing in Mill No. 1, 0.51 kg. in mills Nos. 2 and 3, and 0.41 kg. in mill No. 4, per minute. A screen sizing-test of the heads and tailing or charge fed and product of regrinding of these tube-mills showed the following results:

Tube-mill		Per cent on screen +40	+60	+80	+100	+120	+150	+200	—200
No. 1	Heads.	0.2	5.0	9.8	16.5	5.5	20.0	18.0	24.2
	Tailing	0.0	0.1	1.1	8.0	2.0	16.0	15.7	55.5
No. 2	Heads.	0.3	5.2	9.7	17.2	5.6	20.4	17.7	22.5
	Tailing	0.0	0.5	3.7	11.9	3.2	19.0	17.1	43.4
No. 3	Heads.	0.2	5.3	9.7	17.4	5.4	20.1	17.8	23.0
	Tailing	0.0	0.6	4.1	12.0	3.5	19.5	17.3	41.9
No. 4	Heads.	0.3	5.4	10.0	18.0	6.1	23.5	17.0	18.0
	Tailing	0.0	0.2	2.5	10.5	2.6	17.6	15.1	49.1

In my opinion the marked superiority of the No. 1 mill is due to the larger discharge-opening with which it is supplied, as in other plants I have seen an increase of 20% in the output of a tube-mill obtained by replacing the discharge-end casting of a mill which had a small opening with a casting having an opening of larger size. The products from the regrinding in the tube-mills are elevated by the second belt elevator, already mentioned, and returned to the Dorr classifiers, and thence follow the course already described until all of the pulp has been converted into slime, which flows to the 13 dewatering cones already mentioned, and thence to the 26 Johnston concentrators, for the extraction of such concentrate as they may contain. A screen sizing-test of the heads and tailing of the Johnston concentrators, together with the assay of each separate screen-product, gave the following results:

	+150 Screen Weight Per cent	Silver Kg.	+200 Screen Weight Per cent	Silver Kg.	—200 Screen Weight Per cent	Silver Kg.
Heads	5.21	0.358	5.2	0.370	85.0	0.892
Tailing	5.61	0.332	5.0	0.330	84.1	0.628

The concentrates from all of the tables, both Wilfleys and Johnstons, pass through pipes to a collecting tank located in the concentrate room, where they are drawn out with hoes, air dried and sampled for sale to the ore buyers. The analyses of the concentrate average as follows:

	Per cent.
Lead	2.2
Copper	0.38
Silica	30.0
Iron	23.4
Manganese	3.8
Lime	2.8
Zinc	6.2
Sulphur	13.7
Silver	17.216 kg. per ton
Gold	74 gm. per ton

Cyanidation.—All of the tailings from the 26 Johnston concentrators flow to one of the 24 agitation tanks for treatment. These tanks shown in Fig. 12 are 30 ft. in diameter by 10 ft. deep arranged with mechanical agitators, pumps for agitating and transferring the charge and piping for decanting the solutions and for transferring the slime to the filters. The operations which take place in these tanks during the treatment and the time occupied during each operation are as follows:

	Hours
Filling the tank with its charge of slime	4
Settlement and decantation of excess solution	7 to 9
Adding first solution	1
First agitation, by mechanical stirring and pump simultaneously	20
Settling first solution	13 to 15
Decanting first solution	2 to 3
Adding second solution	1
Second gyratory agitation	6 to 9
Second agitation by compressed air simultaneously	2 to 9
Settling second solution	9 to 11
Decanting second solution	2¾
Adding third solution	1
Third gyratory agitation	2
Settling third solution	11 to 20
Decanting third solution	3
Adding fourth solution	1
Fourth gyratory agitation	1
Settling fourth solution	10
Decanting fourth solution	2½
Discharging tank	1

As the slime is being charged into the tank 2½ cans of burned lime are thrown into the tank to assist the slime in settling. Each can contains about 20 kg. of lime. With the first solution 75 kg. of dry sodium cyanide, 10 kg. of lead acetate and another ½ can of lime are added, and with the third solution another ½ can of lime is added. The tanks generally contain from 55 to 65 tons of dry slime during treatment with 3 tons of solution per ton of dry slime. During the decantations from 3 to 4 ft. in depth of clear solution is decanted off from above the settled slime, and the tank is then refilled with the same quantity of solution from the slime-filter, with or without previous precipitation of its metals as may be convenient. The solution used for this purpose contains 0.25% KCN and 1.1% CaO.

It is customary to assay the slime and solutions at the end of each operation in order to know the extraction obtained at each stage of the operations, and whether 3 or 4 treatments with new solution are given depends on the results obtained, for it may be found that after a certain extraction has been obtained, the value extracted by a longer treatment would not pay the expense of such treatment. Such a state of affairs is shown in the treatment tests,

whose details appear in Tables III and IV following, where it is shown that the amount of metal extracted by the last solutions are relatively small.

Filtration.—For the filtration of the slime in this mill a Butters filter of 100 leaves is used, and each operation of the filter, making cakes ¾ in. thick, requires 1½ hours, in which time from 30 to 40 cubic metres of slime, containing 22 tons of dry slime, are filtered. The distribution of the time occupied in this operation is as follows:

	Minutes
Filling the filter-tank with slime	13
Applying vacuum to the leaves, forming ¾-inch cake	13
Returning excess of slime to stock-tank	12
Filling the filter-tank with wash water	10
Application of vacuum drawing wash-water through the leaves	40
Discharging washed slime to the river	5

As the pores of the filter-cloth become clogged with an incrustation of lime, it is customary to wash 8 leaves every day, in a bath of hydrochloric acid, using 35 litres of acid per day for this purpose. Calculating by difference in volume and by the specific gravity of the filter-cake, it is found that 69 tons of poor solution are discharged with each discharge of the filter, which represents a loss of metal dissolved in such solution of 158.7 gm. of silver and 2.07 gm. of gold in each filtration, or 2.3 gm. of silver and 0.03 gm. of gold per ton of weak solution discharged to the river with the slime-tailing.

Tests in treatment by cyanidation.—In the tables Nos. III and IV are shown the details of two treatment tests made in the Loreto mill on Feb. 24, 1909. These tests were made under identical conditions, allowing the tailing from the Johnston concentrators to flow in a continuous stream for 7½ hours, and dividing this stream by a board placed in the centre of the trough, so that each tank would receive one-half of the stream. The excess solution was decanted off in each case to the centre line of rivets in the tank, and using for the first and second washes in each tank an unprecipitated solution containing 22.5 gm. of silver and 0.18 gm. of gold, and for the other washes precipitated solution. The agitation was made in each test in the ordinary manner, and 65 kg. of sodium cyanide were added to each tank with the first wash besides the usual reagents. All samples were well washed with clean water before assaying, with the exception of the samples marked (a) which were assayed without previously washing with water.

TABLE NO. III.

Tests in treatment by cyanidation at Loreto Mill.

Charge No. 275. 57.19 tons of dry slime. Tank No. 5.

	Sample No.	Assay of slime		Analysis of solution			
		Silver kg.	Gold gm.	Silver gm.	Gold gm.	KCN %	Alkali %
Tank heads	I	0.522	1.00	35.8	0.55	0.24	1.04
Beginning with first solution wash	II	0.490	0.80	42.	0.70	0.25	0.96
First wash after 4 hours	III	0.386	0.80	70.8	0.60	0.29	1.22
First wash after 8 hours	IV	0.344	0.80	95.	0.70	0.26	1.22
First wash after 12 hours	V	0.258	0.50	111.	0.65	0.26	1.22
First wash after 16 hours	VI	0.226	0.50	130.9	0.70	0.26	1.22
First wash after 20 hours	VII	0.192	0.50	147.9	0.75	0.25	1.22
First wash after 24 hours	VIII	0.160	0.50	162.3	0.75	0.24	1.22
After adding second wash of solution	IX	0.127	0.50	71.1	0.35	0.26	1.22
End of second wash with agitation of 8 hours	X	0.112	0.25	79.6	0.35	0.25	1.22
After adding third wash of solution	XI	0.095	0.25	18.5	0.10	0.25	1.12
End of third wash with agitation of 4 hours	XII	0.085	0.25	38.9	0.25	0.24	1.12
After adding fourth wash of solution	XIII	0.085	0.25	21.8	0.15	0.26	1.12
End of fourth wash with agitation of 2 hours	XIV	0.080	0.25	22.4	0.15	0.25	1.12
After adding fifth wash of solution	XV	0.074	0.25	14.5	0.15	0.25	1.12
End of fifth wash with agitation of 1 hour	XVI	0.074	0.25	14.8	0.15	0.24	1.12
Slime delivered to filter	XVII	0.068	0.25	......	......	......	......
Same without washing sample	XVII (a)	0.070	0.25	......	......	......	......
Filter—cake first formed	XVIII	0.070	0.25	16.5	0.13	0.24	1.12
Same without washing sample	XVIII (a)	0.077	0.25	......	......	......	......
Filter—cake after water wash	XIX	0.066	0.25	20.3	0.15	0.19	0.65
Same without washing sample	XIX (a)	0.069	0.25	......	......	......	......
Filter—cake when discharged	XX	0.067	0.25	2.3	0.03	0.02	......
Same without washing sample	XX (a)	0.068	0.25	......	......	......	......

TABLE No. IV.

Tests in treatment by cyanidation at Loreto Mill.

Charge No. 276. 54.16 tons of dry slime. Tank No. 26.

	Sample No.	Assay of slime		Analysis of solution			
		Silver kg.	Gold gm.	Silver gm.	Gold gm.	KCN %	Alkali %
Tank heads	I	0.520	1.00	38.3	0.75	0.24	1.04
Beginning with first wash of solution	II	0.477	1.00	39.3	0.80	0.25	0.96
First wash after 4 hours	III	0.401	0.80	63.5	0.60	0.28	1.22
First wash after 8 hours	IV	0.348	0.80	85.3	0.70	0.25	1.22
First wash after 12 hours	V	0.263	0.50	103.9	0.65	0.27	1.22
First wash after 16 hours	VI	0.230	0.50	124.2	0.65	0.25	1.22
First wash after 20 hours	VII	0.194	0.50	137.6	0.70	0.24	1.22
First wash after 24 hours	VIII	0.160	0.50	149.3	0.70	0.23	1:22
After adding second wash of solution	IX	0.120	0.25	70.	0.30	0.25	1.12
End of second wash with agitation of 8 hours	X	0.102	0.25	72.	0.35	0.24	1.12
After adding third wash of solution	XI	0.090	0.25	35.5	0.20	0.26	1.12
End of third wash with agitation of 4 hours	XII	0.084	0.25	36.5	0.20	0.24	1.12
After adding fourth wash of solution	XIII	0.079	0.25	21.3	0.15	0.25	1.12
End of fourth wash with agitation of 2 hours	XIV	0.082	0.25	20.	0.15	0.23	1.12
After adding fifth wash of solution	XV	0.075	0.25	14.5	0.15	0.24	1.12
End of fifth wash with agitation of 1 hour	XVI	0.075	0.25	13.5	0.15	0.23	1.12
Slime delivered to filter	XVII	0.074	0.25	......	......	......	......
Same without washing sample	XVII (a)	0.072	0.25	......	......	......	......
Filter-cake first formed	XVIII	0.069	0.25	16.3	0.13	0.24	0.96
Same without washing sample	XVIII (a)	0.078	0.25	......	......	......	......
Filter-cake after water wash	XIX	0.066	0.25	20.3	0.15	0.12	0.40
Same without washing sample	XIX (a)	0.069	0.25	......	......	......	......
Filter-cake when discharged	XX	0.066	0.25	2.3	0.03	0.02	......
Same without washing sample	XX (a)	0.068	0.25				

Screen sizing-tests, with assay of each separate screen product, of the samples in Table No. III:

Number of the sample	+150 Screen			+200 Screen			—200 Screen		
	Weight %	Silver kg.	Gold gm.	Weight %	Silver kg.	Gold gm.	Weight %	Silver kg.	Gold gm.
I	7.8	0.296	1.5	7.5	0.300	1.5	84.7	0.560	......
VIII	7.8	0.170	1.6	7.5	0.100	0.5	84.7	0.148	......
X	7.5	0.129	0.5	8.5	0.119	0.5	84.0	0.110	......
XII	8.2	0.120	0.5	7.8	0.090	0.3	84.0	0.085	......
XIV	8.0	0.093	0.3	7.5	0.084	0.3	84.5	0.079	......
XVII	8.5	0.070	0.3	8.8	0.076	0.3	82.7	0.074	......
XX	8.5	0.084	0.3	7.8	0.075	0.3	83.7	0.066	......

Screen sizing-tests, with assay of each separate screen product, of the samples in Table No. IV:

Number of the sample	+150 Screen			+200 Screen			—200 Screen		
	Weight %	Silver kg.	Gold gm.	Weight %	Silver kg.	Gold gm.	Weight %	Silver kg.	Gold gm.
I	8.0	0.300	0.5	7.8	0.300	0.5	84.2	0.560	1.0
VIII	8.0	0.166	0.5	7.8	0.164	0.5	84.2	0.159	1.0
X	8.3	0.120	0.5	8.5	0.117	0.5	83.2	0.107	0.5
XII	7.8	0.095	0.3	7.5	0.085	0.3	84.7	0.084	0.3
XIV	7.8	0.097	0.3	8.9	0.084	0.3	83.3	0.082	0.3
XVII	8.2	0.103	0.3	8.5	0.090	0.3	83.3	0.083	0.3
XX	7.5	0.094	0.3	8.7	0.079	0.3	83.8	0.075	0.3

Precipitation.—The solutions decanted from the agitation tanks contain on an average 96.3 gm. of silver and 0.65 gm. of gold per ton of solution, and after passing through the zinc boxes only contain 4 gm. of silver and 0.1 gm. of gold per ton. The solution, contained in the slime delivered to the Butters filter, which is separated from the slime by passing through the filter leaves, contains on an average 22.5 gm. of silver and 0.18 gm. of gold. This solution is used in the batteries and as a first wash in the agitation tanks, so that it may be enriched before precipitation. The last solution passing through the filter leaves, which has been impoverished by the water wash given to the slime-cake on the leaves, contains about 8 gm. of silver and 0.15 gm. of gold, and is also used in the batteries.

In the precipitation-room there are 20 zinc-boxes of 5 compartments each. These compartments are 3 by 4 by 2 ft. and have a false bottom 6 in. above the bottom of the compartments, thus obtaining a depth of 18 in. for the zinc-shavings, or a contents of 90

cu. ft. of zinc shavings in each zinc-box. Of the 20 zinc-boxes only 13 are required to be used for the precipitation of 1,200 tons of solution daily, or 1 cu. ft. of zinc-shavings per ton of solution precipitated. The zinc-shorts are placed in trays in the first compartment of the boxes for dissolution, although at times, when the quantity of shorts is great, they are treated with sulphuric acid to dissolve the zinc before melting. The precipitate-fines are passed through a filter-press, and the moist cakes of pressed precipitate are dropped through a 12-in. iron pipe into a bin in the melting-room immediately below.

Melting.—The precipitate is taken from this bin and placed in sheet-iron pans 25 by 36 in., which are then placed in muffles 27 by 42 in., heated by a wood fire to such a temperature that the hand may be held in them, where they remain until the precipitate contains but 3 or 4% of moisture. As soon as the precipitate is dry it is mixed with the following flux, calculating the percentages on the dry weight of the precipitate:

	Per cent
Precipitate	100
Borax-glass	15
Sodium bicarbonate	12
Quartz sand	2 to 3

and the mixture is kept in wooden bins around the wall of the store-room until it can be melted. The furnaces used for melting consist of 4 Steele-Harvey revolving furnaces for No. 300 graphite crucibles, heated by oil burners. As soon as a crucible is empty, the oil-valve is closed, the top of the crucible is removed, and the crucible is filled with the mixture of precipitate and flux, using special shovels. The cover is then replaced on the crucible, the oil-valve is opened, and the crucible is allowed to heat until the charge is melted. As soon as the molten charge has settled in the crucible, more of the mixture of precipitate and flux is added until the crucible is again full. It is again heated until the charge is molten and in a state of quiet fusion when the greater portion of the contained slag is poured into slag pots, and the rest of the slag, together with the molten bullion, which remain in the bottom of the crucible, is poured into bullion moulds. The slabs of bullion thus obtained are remelted to obtain clean bars, and the slag is treated as already described in the practice at the Dos Estrellas Mill No. 1, in Chapter IV. The precipitate yields 50% of its weight in bullion assaying 950 fine in silver and from 5 to 7 fine in gold.

Extraction.—During the month of March, 1909, the results of the treatment in the Loreto Mill were as follows:

	Total weight tons	Assay per ton		Total contents		Per cent contents calculated on original ore		
		Silver kg.	Gold gm.	Silver kg.	Gold gm.	Weight %	Silver %	Gold %
Original ore..	9309.437	1.1230	6.19	10454.500	57625	100.00	100.00	100.00
Concentrate.	205.211	18.8540	91.30	3869.000	18742	2.20	37.01	32.52
Tank heads..	9104.226	0.7230	4.27	6585.500	38883	97.80	62.99	67.48
Precipitate...	7.142	617.2580	4042.70	4408.455	28873	....	42.17	50.10
T'lng to river.	9097.084	0.0800	0.30	727.767	2729	....	6.96	4.73
Solution to river (3:1).	27291.00	0.0023	0.03	62.769	818	....	0.60	1.42
Unexplained loss......	....	....	....	1386.509	6463	....	13.26	11.23

As the silver and gold extractions in the preceding table were calculated on the assays of the precipitate and not on the actual bullion produced it is very probable that a considerable part of the "unexplained loss" is not actual loss, but is simply due to the inaccuracy of the sample, as it is impossible to obtain an absolutely correct sample of the precipitate.

Chemical Consumption.—The following quantities of chemicals are consumed per ton of ore treated:

	Kg.
Sodium cyanide................................	1.35
Lead acetate................................	0.20
Burnt lime................................	3.00
Zinc................................	1.20

Cost of treatment.—The cost of treatment per ton of ore is as follows:

Milling and Concentration.

Crushing, sampling and delivery to bins................	₱0.42	
Grinding in batteries and chilean mills..................	0.94	
Grinding in tube-mills..................................	0.40	
Concentration..	0.35	
Superintendence and general expenses....................	0.31	
Total cost of milling and concentration....................		₱2.11

Cyanidation.

Labor..	₱0.63	
Material and supplies..................................	2.13	
Power..	0.26	
General expenses.......................................	0.17	
Total cost of cyanidation...............................		₱3.19

Taxes and Commissions.

Taxes and commissions in selling concentrates and bars, etc..		₱4.81
Total cost of treatment...........................		₱10.11

Power.—Electric power is supplied to this company at the rate of ₱96 per horsepower per year.

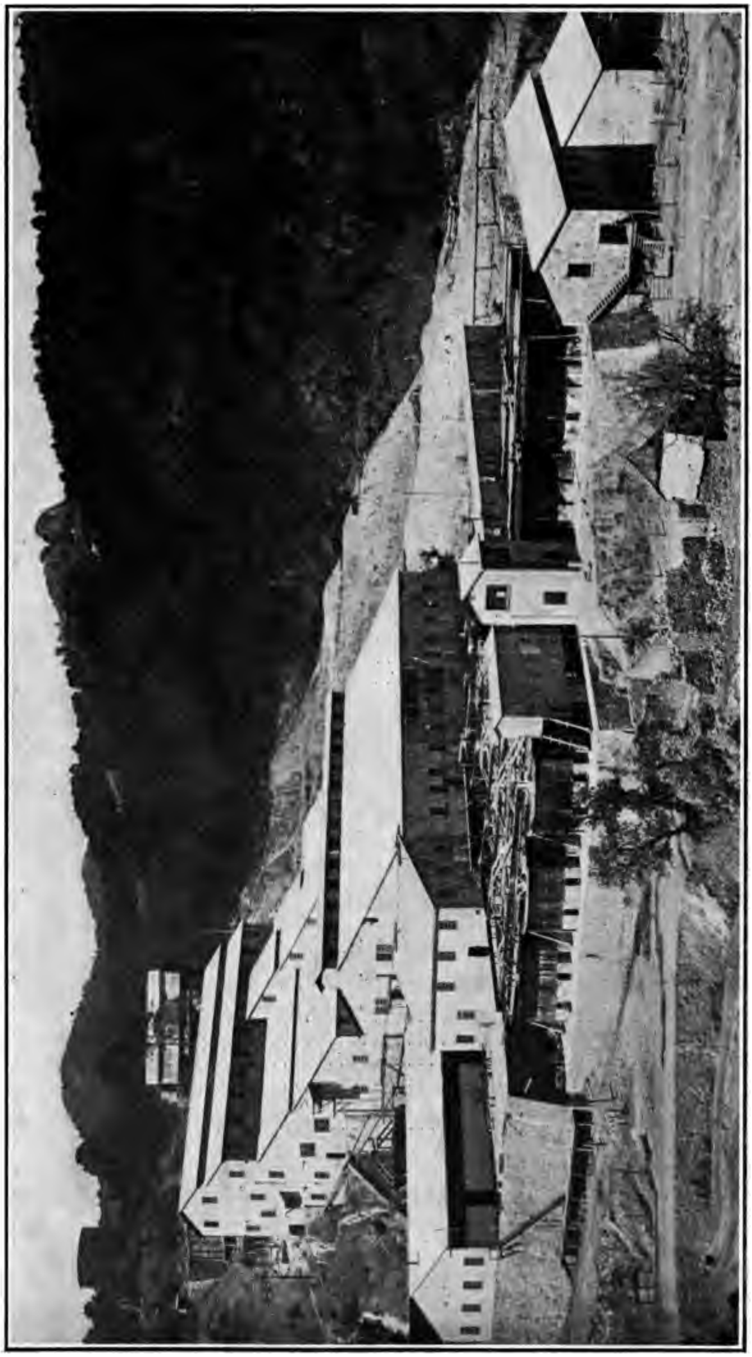


Fig. 14. Guerrero Mill, Real del Monte y Pachuca Company.

Guerrero Mill.

This mill, shown in Fig. 14, is situated in Real del Monte, and has a capacity of 300 tons of ore per day. Its erection, which cost ₱800,000.00, was finished in 1908, and it was at that date the most modern plant that could be constructed for the treatment of silver and gold ores by concentration and cyanidation in flat-bottom tanks with mechanical agitators.

Receiving, crushing and sampling.—The ore from the mines, after being received in the terminal station of the aerial tramway is hauled in small tram cars to the principal storage-bins, which are divided into ten compartments having a capacity of 200 tons each, or a total capacity of 2,000 tons. From these bins the ore is carried to 3 Blake crushers, 9 by 15 in., where it is crushed to the size to pass through a 2-in. ring. After passing through the crushers the ore falls on an endless conveyor-belt which deposits it in any battery-bin that may be empty. These battery-bins have a capacity of 800 tons of ore. As the ore falls from the crushers to the conveyor-belt, it passes through a Snyder sampler which cuts out 1/10 of the ore for the sample. This sample is passed through a set of 14 by 26 in. rolls and thence through another sampler which cuts out 1/10 of this product for the sample. This sample passes through another set of 12 by 20-in. rolls, and then through another sampler which cuts out 1/10 of the product for the final sample which is sent to the assay office. The remainders of the various samples are elevated by a belt-elevator and dropped upon the conveyor-belt which discharges them into the battery-bins.

Milling, classification and concentration.—The battery contains 40 stamps weighing 1,050 lb., which have 106 drops per minute with a 7-in. drop. The stamps grind the ore through a 4-mesh screen, set 2½ in. above the dies, at the rate of 7½ tons per stamp per 24 hours, using from 12 to 13 tons of 0.04% KCN solution per ton of ore ground. The sequence of operations in this mill is graphically illustrated in the flow-sheet shown in Fig. 15, which is explained as follows:

The battery pulp is elevated in a belt-elevator to a sloughing-off cone 6 ft. in diameter whence the slime and excess solution overflow into 14 Callow dewatering-cones 8 ft. in diameter, while the sand discharged from the vertex falls upon 2 duplex Callow screens fitted with 30-mesh screen cloth of No. 33 steel and having a velocity of 100 ft. per minute. The oversize of these screens is fed to 8 Evans-Waddell Chilean mills 6 ft. in diameter having 30-mesh Ton Cap screens, while the undersize flows to 3 hydraulic cones, set in series, whence the slime overflows to the 14 Callow dewatering-cones already mentioned, while the coarse and medium sand discharged respectively from the first and second hydraulic cones run to 4 and to 8 Wilfley concentrating tables, from which the tailing are returned to the Chilean mills already mentioned, by means of

troughs and a belt-elevator which raises them to the necessary elevation. The trough receiving the tailing from the Wilfleys has but 3% grade, and is shaken by machinery in order to cause the tailing to run in it. The fine sand discharged from the vertex of the third hydraulic cone is distributed among 12 Johnston concentrators.

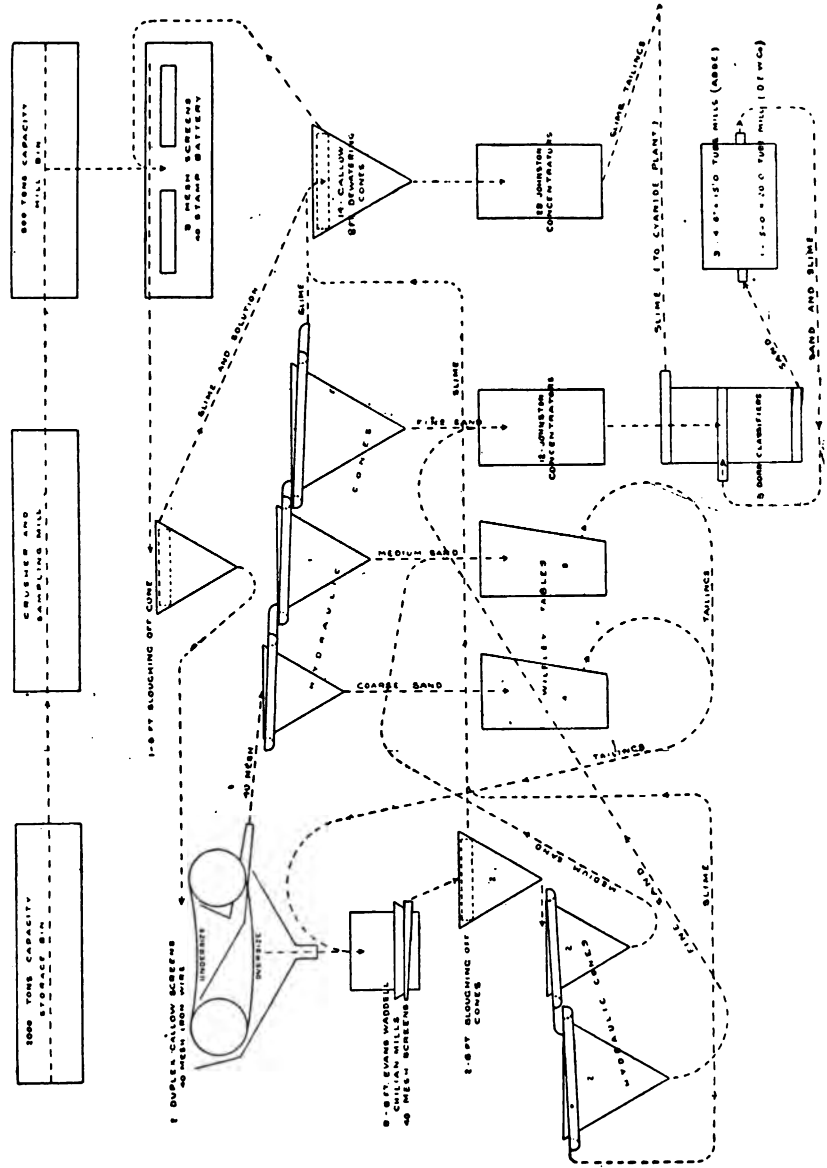


Fig. 15. Flow Sheet of the Guerrero Mill, Real del Monte y Pachuca.

The Chilean mills, which receive the oversize from the Callow screens and the tailing from the Wilfleys, run 30 r.p.m., and it is interesting to note that, notwithstanding the fact that a 30-mesh Ton Cap screen is used in these mills, but 4% of the product remains on a 40-mesh screen, while 32.4% passes through a 200-mesh screen. The product of the Chilean mills flows to 2 sloughing-off cones 6 ft. in diameter, whence the slime overflows to the same 14 Callow dewatering cones previously mentioned, while the sand discharged from the vertices flows to 2 hydraulic cones 4 ft. in diameter, from which the medium-sized sand discharged at the vertices flows to the 8 Wilfley tables already mentioned as handling medium sand, while the fine sand and slime, which overflow, fall into 2 hydraulic cones 6 ft. in diameter from which the fine sand discharged from the vertices runs to the 12 Johnston tables already mentioned while the slime flows to the 14 Callow dewatering cones previously mentioned. The tailing from the 12 Johnston concentrators runs to 3 Dorr classifiers from which the slime passes to the agitation tanks in the cyanidation plant, while the sand, containing 50% of moisture, passes to 4 tube-mills for regrinding, adding 1.2 kg. of lime to the mills per ton of ore entering in them.

Three of these mills are Abbé mills 4½ ft. in diameter by 15 ft. long, while the other is a Denver Engineering Works mill 5 ft. in diameter by 20 ft. long. The reground product from these four mills is elevated by a belt-elevator and returned to the Dorr classifiers and thence follows the course previously mentioned until all of the sand has been converted into slime. From the fourteen 8-ft. Callow dewatering cones the solution which overflows runs to the stock-tank for battery solution, while the slime discharged from the vertices flows to 28 Johnston concentrators the tailing from which passes to the agitation-tanks of the cyanidation plant.

The following screen analysis of the various mill-products was taken at the Guerrero mill during the month of July, 1909. A study of the figures will show some interesting features and special attention might be called to the work of the Callow screens which at 100 ft. belt-travel per minute (a very high speed) when equipped with 30-mesh No. 33 steel wire (the purpose being to separate as undersize all material that can be made to pass a 40-mesh laboratory screen) gives an oversize 92% of which remains on a 40-mesh screen and 96.5% remains on 60-mesh screen. At the same time an undersize is sent to the hydraulic cones, 9½% of which passes a 40-mesh, said product containing practically all of the liberated mineral. The effect of this system is seen in the enormously wide streak of mineral on the first three Wilfley tables, which really receive as feed a product that has once been concentrated by the Callow screens and once by the hydraulic cones following these.

Another feature of interest to concentrator-men, is the analysis of the product of the Chilean mills, 4% of which remains on a 40-

mesh screen and 32.4% of which passes a 200-mesh screen, notwithstanding that a 30-mesh Ton Cap screen is used in the mills. The discharge through these screens is kept up to the maximum, a heavy splash being used, but all material coarser than 60-mesh is, so far as possible, cut out in the first hydraulic classifiers following these mills and sent back to the Wilfley tables, where it joins the oversize from the Callow screens and returns again to the Chilean mills. Another feature of the grinding at Guerrero mill is the fineness of the tube-mill feed, only 7% of which remains on a 60-mesh screen. This is of course absolutely against high tube-mill tonnage, but is justified by the necessity for the most careful possible concentration, which means on these ores, pretty fine grinding (in stages) and careful elimination of mineral after each step and before the tube-mills get a chance to slime the mineral. The apparently low efficiency of the tube-mills receiving such feed (in comparison with, for instance the Loreto mill of the same company, in which a much simpler flow-sheet is in use) is principally due to the fact that all of the easily slimed portion of the ore has been made to pass a 200-mesh screen during its repeated passage through the Waddell mills, leaving for the tube-mills nothing but the most refractory portion to grind.

TABLE No. V.

Screen Analysis of Samples Taken at Guerrero Mill.

		Solids Per cent	PERCENTAGE ON SCREENS									
			on10	on20	on40	on60	on80	on100	on120	on150	on200	—200
Battery discharge	....	20.74	36.0	20.0	16.4	6.8	2.6	3.6	0.5	2.0	1.8	10.3
Elevator No. 1—Sloughing-off-cone	Overflow	3.59	....	....	....	....	....	0.5	0.1	2.5	4.9	92.0
	Spigot	30.85	35.8	20.1	17.0	7.5	3.0	3.9	0.8	2.5	2.0	7.4
Callow screens	Oversize	35.50	43.0	27.0	22.0	4.5	1.5	0.5	....	0.3	0.1	1.1
	Undersize	11.92	....	....	8.8	23.7	12.0	14.7	2.0	7.0	6.0	25.8
Wilfleys.pulp from Callows	Feed	18.00	....	....	8.7	28.0	14.8	16.3	2.9	7.8	5.5	16.0
	Discharge	9.33	....	....	8.6	28.1	15.0	16.2	2.8	7.5	5.9	15.9
Wilfleys. Chilean return	Feed	20.35	....	....	11.5	44.0	16.3	12.3	2.5	4.2	2.5	6.7
	Discharge	11.37	....	....	12.0	44.0	16.0	12.0	2.5	4.1	2.6	6.8
Wilfley tables general	Feed	18.94	....	....	11.6	37.0	15.0	14.7	1.6	5.1	3.9	11.0
	Discharge	11.87	....	....	11.7	36.9	15.2	14.8	1.5	5.0	4.0	10.9
Waddell mills	Feed	22.46	19.5	11.4	16.0	24.5	9.6	8.2	1.0	4.0	2.2	3.6
	Discharge	22.60	....	....	4.0	23.8	12.7	14.5	1.2	6.8	4.6	32.4
Elevator No. 3—Sloughing-off-cone	Overflow	4.41	....	....	....	0.2	0.4	0.5	0.1	0.8	1.5	96.5
	Spigot	25.93	....	....	6.4	28.5	13.2	14.5	2.0	8.2	4.5	22.7
Elevator No. 4—Sloughing-off-cone	Overflow	4.49	....	....	....	0.3	0.5	0.4	0.1	1.5	3.7	93.5
	Spigot	27.58	....	....	7.5	29.6	13.0	15.9	2.5	8.8	3.5	19.2
60-inch No. 1 classifier	Overflow	5.59	....	....	....	0.8	0.9	1.0	0.5	2.6	3.3	90.9
	Spigot	17.90	....	....	0.6	12.0	10.5	18.7	4.8	10.0	8.0	35.4
60-inch No. 2 classifier	Overflow	6.09	....	....	....	0.9	1.0	1.1	0.5	2.5	4.0	90.0
	Spigot	24.02	....	....	0.5	14.3	12.9	21.4	2.7	12.9	9.8	25.5
Johnston slime tables	Feed	13.52	....	....	....	0.1	0.2	0.4	0.1	2.5	3.2	93.5
	Discharge	9.53	....	....	....	0.1	0.3	0.3	0.1	2.6	3.2	93.4
Johnston fine sand tables	Feed	18.75	....	....	0.5	13.2	11.5	20.4	3.0	12.2	10.5	28.7
	Discharge	14.21	....	....	0.4	13.1	11.3	20.3	3.1	12.1	10.7	29.0
Dorr classifiers	Feed	32.94	....	....	0.5	6.9	8.3	22.5	3.7	18.8	13.1	26.2
	Slime o'rflow.	12.15	....	....	....	....	0.1	2.5	1.0	9.5	11.0	75.9
	Sand disch'ge	60.47	....	....	0.5	7.7	26.0	5.0	23.8	13.5	13.5	14.3
Tubemill No. 1, 4½x15 ft.. Abbé	Feed	59.76	....	....	0.5	7.3	11.0	25.0	4.5	23.5	13.1	15.0
	Discharge	60.90	....	....	....	3.3	6.5	21.7	4.7	22.6	13.5	27.7
Tubemill No. 2—4½x15 ft. Abbé,	Feed	60.00	....	....	0.4	7.7	10.7	27.8	5.0	20.8	12.8	14.8
	Discharge	61.32	....	....	....	2.3	5.5	20.6	4.7	20.0	15.0	31.9
Tubemill No. 3—4½x15 ft., Abbé	Feed	58.64	....	....	0.5	7.5	11.9	26.5	4.4	22.0	12.5	14.7
	Discharge	59.34	....	....	....	3.7	6.7	23.2	3.1	21.2	14.2	27.9
Tubemill No. 4—5x20 ft., D.E.WC.	Feed	59.84	....	....	0.5	8.5	10.0	26.0	4.5	24.1	12.8	13.6
	Discharge	60.87	....	....	....	3.1	7.2	22.7	4.7	23.0	12.5	26.8
General	Slime to tanks	10.60	....	....	....	....	....	2.0	0.2	6.6	8.1	83.1

The belt-elevators in this mill consist of rubber belts 16 in. wide, running at a velocity of 375 ft. per minute, to which are attached cast-iron buckets 14 by 8 in. The concentrates from the various concentrating-tables flow through pipes to a tank in the concentrate-room, from which tank they are drawn out and spread on the cement floor to dry. These concentrates contain on an average 40% of the values contained in the original ore. The analysis of the original ore and of the concentrate on general samples taken in November, 1908, gave the following results:

	Original ore Per cent	Concentrate Per cent
Copper	Traces	0.73
Iron	2.82	....
Manganese	3.58	....
Iron and manganese	....	22.76
Lime	1.55	0.91
Zinc	2.22	11.55
Sulphur	1.56	20.47
Insoluble	79.02	21.95
Lead	Traces	1.50
Silver, assay in Kg. per metric ton	0.849	8.584
Gold, assay in Gm. per metric ton	4.24	36.35

The ore treated during the month of March, 1909, was of lower grade, containing but 0.561 kg. of silver and 2.70 gm. of gold per ton.

Cyanidation.—By this system of milling and classification all of the ore is converted into slime of such a fineness that 83.1% will pass through a 200-mesh screen. This slime is collected in one of the agitation-tanks of the cyanidation plant, from which the excess solution is decanted while the slime is entering until the tank contains a charge of 50 tons of dry slime. As soon as this occurs the slime-flow is diverted into another agitation-tank for collection while the charge in the first tank is being treated. After the charge in the tank has settled, the excess solution is decanted off, and the tank is filled up with the first wash of cyanide solution containing 0.15% KCN, together with a sufficient quantity of dry sodium cyanide to raise the strength of the solution in the tank up to 0.2% KCN; 10 kg. of lead acetate are also added to the tank. The charge is then agitated for 24 hours, and then allowed to settle and the supernatant solution decanted off, the operations of settlement and decantation requiring about 3½ hours. A second wash is given to the charge by filling up the tank with 0.15% KCN solution and agitating for 8 hours. The solutions from the first and second washes flow directly to the precipitation-boxes. A third and fourth wash are given to the charge, using in each case solution containing 0.15% KCN, with 8 hours of agitation, and the solutions decanted from these two washes flow to a storage tank for use as first and second washes in other agitation-tanks. The total time required for collecting a charge in a tank and giving it the four washes just described is ordinarily four days. The proportion of solution to

dry slime in a tank during treatment is 3 to 1, or 150 tons of solution to 50 tons of dry slime. After decanting the fourth solution the slime remaining in the tank is transferred to a Butters filter where the remaining solution is separated out. The filter cakes are given a wash with clean water to remove any solution which might remain in them, and are then discharged to the river. This Butters filter contains 100 filter leaves, each of which filters three tons of dry slime per day.

Tank equipment.—The cyanidation plant contains the following tank equipment:

Twenty-seven agitation-tanks, each 30 ft. in diameter by 10 ft. deep, equipped with mechanical agitators and Traylor centrifugal pumps for transferring the solution and slime.

One storage tank 30 ft. in diameter by 15 ft. deep for storing slime ready for the filter.

Ten solution tanks, each 30 ft. in diameter by 10 ft. deep, of which 3 are used for clarifying solutions, 2 for decanted solutions, 2 for precipitated solutions, 2 for storing solutions and 1 for wash solutions from the Butters filter.

One water tank 30 ft. in diameter by 30 ft. deep for the water used in milling.

Extraction.—There is a slight variation between the results obtained in the treatment of each tank, and although I was informed that in general the extraction varied between 80 and 85% of the silver and gold content of the ore, the results obtained on the day preceding my visit to this mill were much better, being as follows:

	Silver Kg. per ton	Gold Gm. per ton
Battery pulp assayed	0.800	5.00
Concentrate assayed	12.210	54.00
Slime charged to agitation-tanks	0.390	1.00
Tailing from Butters filters	0.080	0.25

The above assays show that 51.3% of the silver and 80% of the gold was extracted from the original ore by concentration and 38.7% of the silver and 15% of the gold contained in the original ore was extracted by cyanidation, giving together a total extraction of 90% of the silver and 95% of the gold. A further examination of the above assays shows that 79.5% of the silver and 75% of the gold contained in the slime fed to the agitation-tanks was extracted by cyanidation in these tanks, but as the total results are what count in calculating the profits derived by any treatment it will be observed that the extraction on the original ore is satisfactory.

Precipitation.—In the precipitation-house there are 14 zinc-boxes of 5 compartments each. These compartments measure 3 by 4 by 2 ft. and have a false bottom 6 in. above the bottom of the compartments, so that the depth of the filling of zinc-shavings is 18 in. The shavings are cut to a thickness of 1/300 of an inch and 1 cu. ft. of

zinc-shavings is present for each 1.5 tons of solution precipitated daily. The solution entering the zinc-boxes assays 54 gm. of silver and 0.35 gm. of gold per ton of solution. The solutions after precipitation, upon leaving the zinc-boxes, assay from 0.1 to 1 gm. of silver and traces of gold per ton. The precipitate from these zinc-boxes yields upon melting 55% of its weight in bars assaying 935 fine in silver and 5 fine in gold. The precipitate is sent in sealed iron cans to the Loreto mill of the same company for melting.

Consumption of chemicals.—The following chemicals are consumed in this mill per ton of ore treated:

	Kg.
Sodium cyanide containing 128 per cnet	1.25
Zinc	0.6
Lime	4.5
Lead acetate	0.2

Cost of treatment.—The cost of treatment in this mill is as follows:

Milling and Concentration.

Crushing, sampling and delivery to bins	₱0.31	
Crushing in stamp batteries	0.30	
Concentration on Wilfley concentrators	0.09	
Grinding in Chilean mills	0.44	
Grinding in tube-mills	0.38	
Concentration on Johnston concentrators	0.23	
General expenses and superintendence	0.43	
Total cost of milling and concentration		₱2.18

Cyanidation.

Pumping	₱0.12	
Sodium cyanide	1.11	
Lead acetate	0.12	
Lime	0.08	
Operating expense	0.19	
Repairs	0.04	
Cost of filtering slime in the Butters filter	0.11	
Total cost of cyanidation		₱1.77

Precipitation.

Zinc	₱0.31	
Operating expense	0.14	
General expense	0.18	
Superintendence	0.08	
Total cost of precipitation		₱0.71
Taxes and smelting charges on concentrate and bars		3.43
Total cost of treatment		₱8.09

CHAPTER IX.

CYANIDE PRACTICE IN THE TAILING PLANT OF THE BLAISDELL COSCOTITLAN SYNDICATE,

Pachuca, Hidalgo.

The following description was supplied in 1909 through the courtesy of Messrs. H. A. Barker, J. R. Brown, and F. H. Jackson, respectively manager, constructing engineer, and cyanide superintendent for this company. This plant was constructed for treating the tailing resulting from the Patio treatment of the ores of this camp during the last 350 years. The tailing deposited on the plain below Pachuca amounts to about two million tons; it covers the ground to a depth of from one to six feet, and is easily collected by scrapers and cars. This old tailing assays on an average of ₱6 per ton, of which amount ₱2.50 represents the value of the gold and ₱3.50 represents the value of the silver contents, estimating the value of 1 gm. of gold at ₱1.329, and 1 gm. of silver at ₱0.032. In order to avoid confusion between the old tailing and the tailing from the cyanidation the former will hereafter be termed the ore. There are two plants, that for agitation and that for percolation; the former having been first erected, while the latter is of more recent age, is much simpler, and requires none of the expensive apparatus in the former.

The agitation plant and treatment.—The old tailing or 'ore' is collected at the rate of 440 tons per day in cars of 1 ton capacity, a train of which, shown in Fig. 16, is hauled by a small locomotive to a bin, excavated below ground level, where the cars are dumped and the ore is mixed with 7½ kg. of burnt lime per ton of ore. Through holes in the bottom of this bin the ore is discharged upon an endless elevator belt, shown in Fig. 16, (bottom), which carries it to two mixing-tanks, placed above the treatment tanks. In these mixing-tanks the ore is mixed with 150% of 0.04% KCN solution fed into each mixer through a 2 in. pipe.

The ore, mixed with lime and cyanide solution, discharges through an opening in the centre of the mixers, passing through a 4-mesh wire screen which retains any roots or sticks contained in the ore. These roots, etc., are removed from the surface of the mixture by a laborer using a shovel made of a piece of 4-mesh screen nailed to a wooden handle. The pulp discharged through the opening in the centre of the mixers flows to one of three agita-

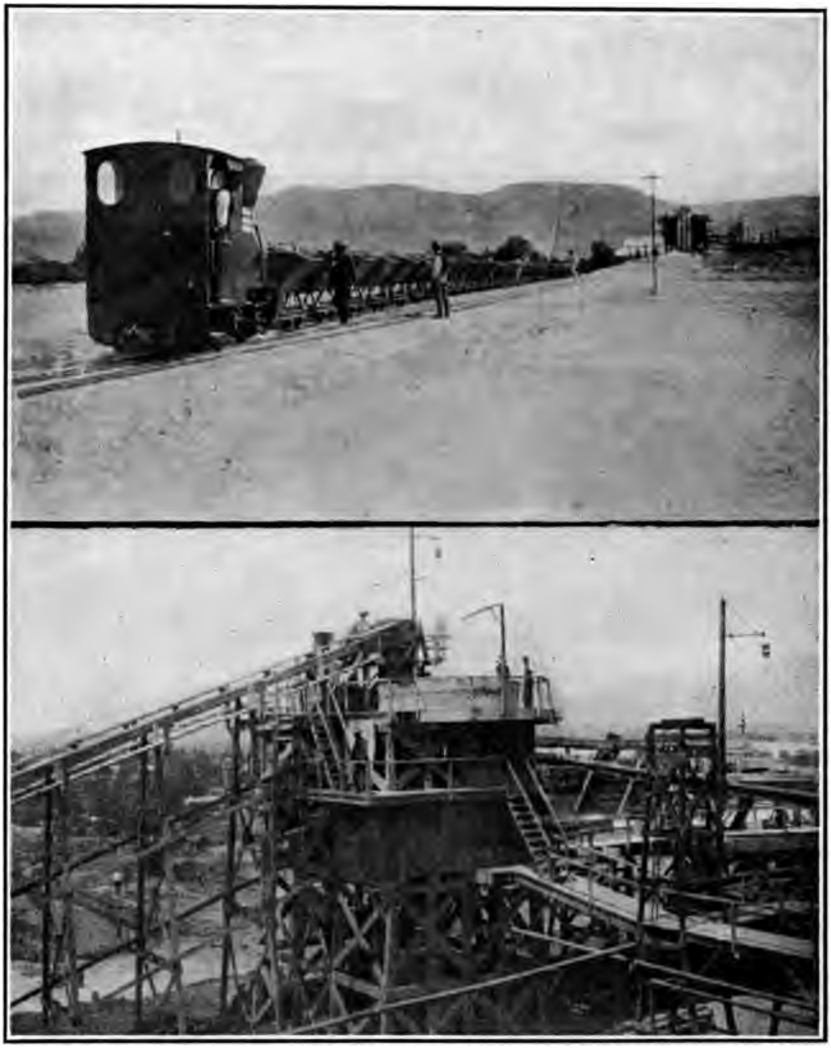


Fig. 16. Coscotitlan Plant; Cars Conveying Tailing; Elevator Belt and Mixing Tanks.

tion-tanks, which are 36 ft. in diameter by 14 ft. deep, arranged with mechanical agitators giving 5½ revolutions per minute. These agitation tanks are shown in Fig. 17, top plate. The ordinary charge in one of these tanks is 107 tons of ore, together with 150 tons of solution, and it usually takes three hours to fill a tank.

As the various cyanicides contained in the ore usually consume all of the cyanide contained in the solution added to the mixers,

from 50 to 75 kg. of dry sodium cyanide is added to each charge in the agitation tanks to raise the cyanide contents of the solution to 0.05% KCN. The treatment in the agitation-tanks consists of 9½ hours' agitation by the mechanical agitators, during which time the pulp is constantly circulated in the same tanks, by 6-in. Morris

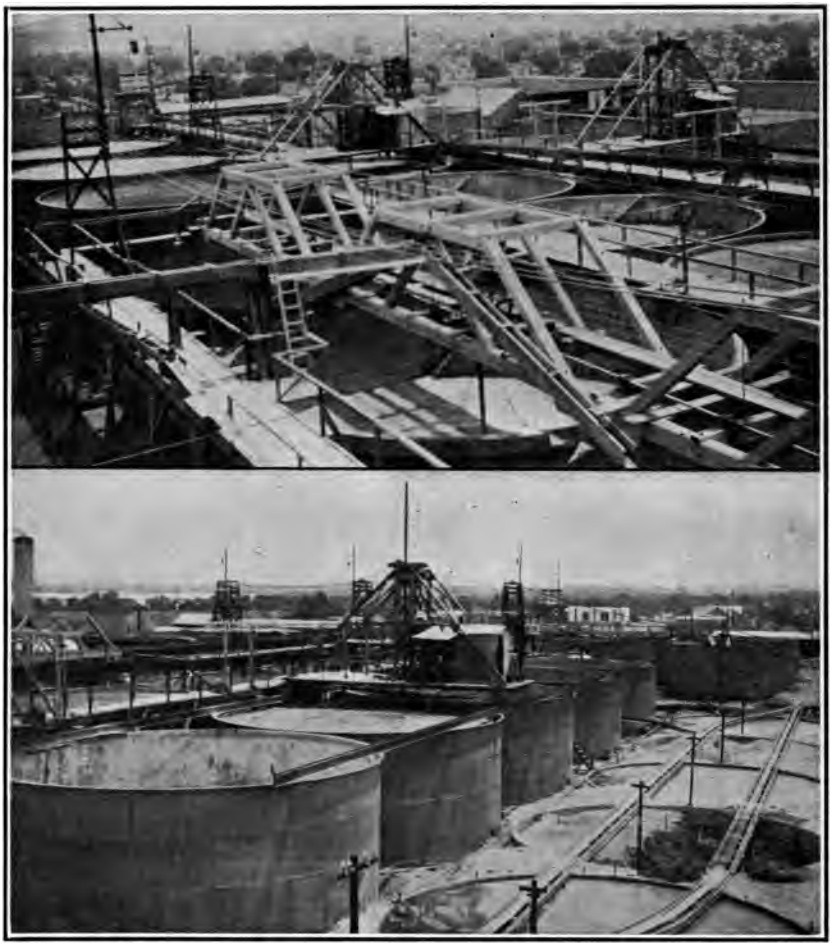


Fig. 17. Coscotitlan Plant; Agitation and Decantation Tanks; Decantation Tanks With Cement Tanks of Percolation Plant Below.

centrifugal pumps, which draw off the pulp from below and discharge it into the tank on top, and by compressed air. The pulp after this treatment is transferred to one of the 10 decantation-tanks, which are of the same size as the agitation-tanks, for further treat-

ment. In these tanks, Fig. 17, bottom part, the charge is given three washes with 0.05% KCN solution, adding 10 kg. of lead acetate to the charge with the first wash. The treatment in these washes consists of one hour of agitation by means of the Blaisdell mechanical agitator, shown in both plates of Fig. 17, which moves on rails above the tanks so that it can be used in any tank for agitation. After the agitation the charge is allowed to settle for 12 hours and the supernatant solution is then decanted off. The tank is then refilled with 0.05% KCN solution, agitated, settled and decanted and these operations are repeated.

The solutions from the first two washes are passed to filter-tanks and thence to the precipitation-boxes. The solution from the third wash is pumped to the tank which supplies the mixers. After decanting the solution from the third wash the treated ore remaining in the tank is discharged into the tailing-pond, shown in Fig. 18, through a 6 in. pipe by means of a Morris centrifugal pump. It takes one hour to discharge a tank into the pond, the dilution of the pulp being 1 of ore to 1½ of solution. The total time of treatment from the entrance of the ore into the mixers to its discharge into the tailing-pond is 72 hours. The value of the gold and silver contents of the treated tailing discharged into the pond is ₱1.90 per ton, and the solution discharged with it contains gold and silver amounting to ₱0.60 per ton.

The tailing is discharged into the pond from a trough, shown in Fig. 18, running around the edges of the pond, and as much of the tailing settles around the edges where first discharged a natural depression is formed in the center of the pond into which the excess solution flows, and is thence conducted to a sump-tank, whence it is pumped into the solution-tanks supplying the mixers.

Extraction.—The treatment in the agitation-plant gives an extraction of 65% of the silver and 90% of the gold, or taken together, an extraction of from 75% to 77% of the metals.

Tank equipment.—Besides the tanks for the treatment of the ore by agitation and decantation already mentioned there are 4 tanks of the same size for storing solutions; one being for precipitated solutions where all of the water required in the treatment is added; another being for solution decanted from the third wash; and the other two being for solutions decanted from the first and second washes which have not yet passed through the precipitation-boxes. There are also 3 filter-tanks, 36 ft. in diameter by 8 ft. deep, through which the solutions pass before entering the storage-tanks just mentioned. These filter-tanks have strips of boards on the bottom on which is stretched first a layer of cocoa-matting and then a covering of burlap, and on top of the burlap is placed a layer 10 in. deep of clean coarse sand.

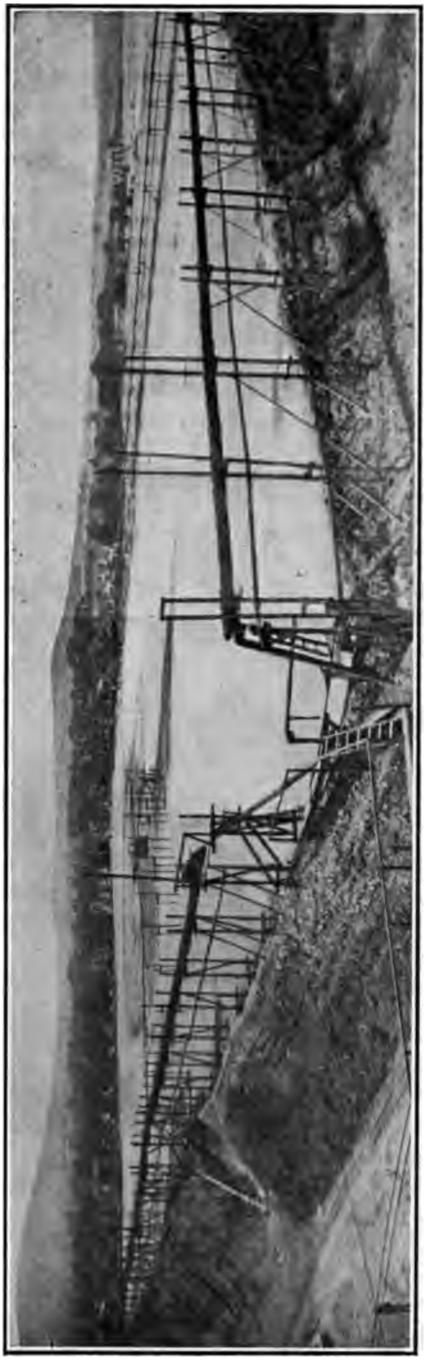


Fig. 18. Coscotitlan Tailing Pond.

Percolation plant and treatment.—In order to determine whether the old patio tailing might be treated by percolation, without the expense for power and costly machinery which is required by the agitation-plant, 5 cement tanks were constructed for the purpose of the test, and the results therein obtained were so favorable that steel tanks having a capacity of 600 tons per day have now been erected. As the construction of the cement tanks and the treatment given in them is very simple, and applicable to small properties in other districts where a simple and cheap plant is desired, the following description may be of interest.

Construction of cement percolation-tanks.—These tanks, shown on the right in Fig. 17, bottom part, were constructed by digging a hole 39 ft. in diameter by 6 ft. deep, within which a wall of stone and mortar 18 in. thick was built around the sides, from the bottom of the hole to the surface of the ground, thus forming a tank 36 ft. in diameter. The bottom of this tank was filled to a depth of 6 in. with stone and mortar, on top of which was laid a layer of thin brick (1 in. thick), laid in mortar, and then the whole of the interior of the tank, bottom and sides, was plastered with a coating ½ in. thick of cement mortar, made by mixing 1 part of good Portland cement with 4 parts of clean sand. The bottom of the tank was slightly inclined towards one side, where an iron pipe 6 in. in diameter was located, through which the solution which had percolated through the ore might be removed, or washings from below given to the charge. Strips of board were placed on the cement bottom on top of which were stretched strips of cocoa matting and then a burlap covering. On top of the burlap strips of 1½ in. board, 6 in. wide, were placed on edge at intervals of 4 in. between boards, and the space between the boards was filled with clean sand. The strips of board serve to protect the sand filter and act as a guide for the shovels when emptying the tank. The tank is now made, and is ready for the charge. It measures 36 ft. in diameter and 4½ ft. in depth, and has a capacity of 125 tons of ore. Two tracks for small cars cross the tank so that it may be easily filled and emptied.

Details of the treatment.—As soon as a tank is filled with ore which has been mixed in the charging cars with 7½ kg. of lime per ton of ore, solution containing 0.04% KCN is introduced underneath the filter bottom and continues entering thus until it shows above the charge. The current is then reversed, allowing the solution to enter on top and discharge below for 4 hours. The introduction of cyanide solution is then stopped, and the charge is allowed to drain and aerate for 2 hours. Then another solution wash is given from above for 4 hours, followed by 2 hours of drainage and aeration. The total treatment consists of 30 washes of 4 hours each, with 2 hours of drainage and aeration between each wash, making 4 washes per day or a total treatment of 8 days. The first wash is given from

below, but all subsequent washings are introduced on top. With each wash after the first, 50 kg. of dry cyanide are added to the solution. The tanks are filled and emptied by hand labor at a contract price of ₱36.00 per tank. The solutions from this treatment are pumped to the same storage-tanks used for solutions from the agitation plant.

Extraction.—The percolation treatment just described extracts 73% of the value of the gold and silver contained in the ore, the extraction of the gold being the same, although the extraction of the silver is a little less than that obtained in the agitation plant. It is believed that the cause of this lower extraction lies in the fact that the old patio tailing treated contained lumps of dried slime which were not broken up in this treatment and consequently were not penetrated by the cyanide solution, as in the new plant of large steel tanks, all of the tailing discharged after treatment, by a stream of water from a hose, passes through a revolving screen which separates out all untreated lumps and delivers them to the agitation plant for further treatment, and it has been found in the new percolation plant that the extraction is equal to that obtained in the agitation plant.

Precipitation.—There are 10 zinc-boxes for the precipitation of the solutions of the Coscotitlan plant, each of which is divided into 5 compartments 3 ft. long by 4 ft. wide by 2 ft. deep, with false bottoms of wire screen. The first four compartments are filled with zinc-shavings, while the last compartment is left empty so that any precipitate which has been carried over into this compartment may settle. Ordinarily 1,600 tons of solution are precipitated daily in these boxes so that there is 1 cubic foot of zinc-shavings for every 2 tons of solution precipitated daily. The zinc-shavings, before being placed in the boxes, are submerged in a solution of lead acetate until they are covered with a gray porous coating of metallic lead, whereby the copper which is precipitated on the shavings does not interfere with the precipitation of the gold and silver. As old patio tailing contains a considerable quantity of copper sulphate, which was introduced during that treatment, it is found that in the precipitation-boxes in this plant the zinc is entirely covered with a red precipitate of metallic copper, but, owing to the porosity of the metallic lead precipitate covering the zinc, the precipitation of the gold and silver is almost perfect. The solutions entering the zinc boxes ordinarily carry silver and gold in value amounting to ₱1.05 per ton of solution, of which ₱0.45 represents the silver and ₱0.60 represents the gold contents. After precipitation the solution leaving the boxes carries but ₱0.02 in values per ton, showing that almost all of the metals have been precipitated. These solutions ordinarily contain 0.04% of CaO, and 0.04% of KCN, and dry sodium cyanide is added to the first compartment so as to raise the strength of the

solution to 0.08% KCN, not only to dissolve a portion of the precipitated copper so that the bars after melting may be finer, but also to dissolve the white precipitate which deposits on the shavings when the solution is too weak.

The zinc-shorts are placed in trays having wire screen bottoms, in one of the compartments. Each tray contains a layer of shorts 2 in. thick and four trays are placed in a compartment, separated from each other by wooden blocks 2 in. thick placed in each corner.

Clean-up and melting.—The zinc-boxes are cleaned up once a week, washing the shavings in each compartment by shaking briskly in the solution until the precipitate and zinc-shorts are detached and fall to the bottom of the compartment. The shavings are then temporarily removed from the compartment, the excess solution is siphoned off and the precipitates and shorts are washed out upon a 50-mesh screen which separates them into the two products. The shorts which are left on the screen are scrubbed with a broom and washed with water while still on the screen before being placed in trays and returned to the zinc-boxes, while the precipitate which passes through the screen runs to a cloth filter tank. The construction of this filter is so simple and inexpensive that it is of special interest to those in out of the way places where transportation is difficult, as it can be easily built on the ground. It consists of a round tank 8 ft. in diameter and 18 in. deep, built of sheet iron riveted together. The bottom of this tank is connected by a 1-in. pipe to a small vacuum pump. On the bottom of the tank several wooden strips are laid, on top of which is stretched a covering of cocoa matting. The interior of the tank is then lined with a covering of cotton cloth, made of strips of ordinary cheap cotton cloth sewed together of such a size that it lines the bottom and sides of the tank and extends 2 in. over the sides. This cloth is burned every month and the ashes added to the precipitate. The precipitate from the zinc-boxes runs into this filter tank and the vacuum pump is started and draws the solution carrying the slime through the filter-cloth, whence it is discharged from the pump into the head of the zinc-boxes, so that any fine precipitate which may have passed through the filter-cloth may settle in the zinc-boxes. As soon as the solution has been sucked through the filter, the moist precipitate is removed and placed in a drying pan, made of sheet iron 9 ft. long by 5 ft. wide by 1 ft. deep, placed on brick pillars and heated by a wood fire underneath. The precipitate is dried in this pan until it contains but 15% of moisture. It is then removed and melted after mixing with the following flux:

	Per cent.
Precipitate	100
Borax-glass	15
Sodium bicarbonate	12
Silica or quartz sand	6

There are 4 melting furnaces in which graphite crucibles No. 300 are used. Two of these furnaces are arranged for coke, while the other two have oil burners. These latter furnaces have the corners filled up so that their shape conforms to that of the crucibles, and the crucibles are placed on a fire brick placed at such an angle to the entrance of the oil that the flame upon striking the brick takes a tangential direction and circulates around the crucible. The melting is conducted as follows: The crucible is filled with the mixture of precipitate and flux, and as this melts and sinks in the crucible more of the mixture is added until the crucible is full of the molten mixture. The heat is then increased until the flux is entirely melted and liquid. The crucible is then removed from the furnace by means of tongs and pulley. The greater portion of the slag is poured off into a slag-pot shaped like an inverted cone, and the balance of the slag and bullion contained in the crucible is then poured into a bullion mould. The slabs of bullion thus obtained are remelted together in another crucible to obtain the bars of bullion which are exported.

The slag-pots have a tap hole, about 3 in. above the bottom, which is stopped up with a clay plug while the pot is filling. As soon as the slag has cooled in the pot sufficient to form a ½ in. crust or shell around the sides the plug is tapped and the interior slag is run out into moulds, a sample being taken of the liquid slag as it flows. The slag from the interior of the pot usually contains gold and silver amounting to $50 per ton, while the pot shells contain values amounting to $1,200 per ton. These shells are remelted to extract the values contained in the shape of buttons of gold and silver bullion. The weekly clean-up ordinarily yields 600 kg. of precipitate which after melting yield 40% of their weight in bars assaying from 750 to 800 fine in silver and 25 fine in gold. The bullion slabs from the first melt ordinarily assay 600 fine in silver, showing that the bars are refined in remelting.

Consumption of chemicals.—The consumption of chemicals per ton of ore treated in this plant is as follows:

	Kg.
Sodium cyanide	1.0
Zinc	0.2
Lead acetate	0.1
Lime	7.5

Of the cyanide consumption 75% occurs in the mixing-tanks where the solution is first attacked by the cyanicides contained in the ore.

Cost of treatment.—During the month of March, 1909, 12,182 tons of old patio tailing were treated at a cost of treatment as below specified. It was impossible to make a perfect separation of the

cost of treatment in the different plants of agitation and percolation, as some parts of the plant were used for both systems of treatment, although it is evident that the treatment by percolation is much the cheaper. The cost has been distributed in two forms as will appear below:

Cost of treatment.

Bin, elevator belt, and mixers	₱0.09
Agitation plant	0.88
Percolation plant	0.11
Pumping	0.13
Precipitation, including cyanide added here	0.39
Melting	0.12
Tailing-pond	0.04
Assays	0.04
Total cost per ton	₱1.80

The cost may also be stated:

Cost of treatment.

Wages and salaries	₱0.29
Renewals and repairs	0.02
Supplies and chemicals	1.28
Electric power	0.17
Electric light	0.04
Total cost of treatment per ton	₱1.80

The above costs do not include the collection of the old patio tailing nor the hauling of it to the bins, which is done by contract at a cost of ₱0.25 per ton.

CHAPTER X.

CYANIDE PRACTICE IN THE SAN FRANCISCO MILL NO. 1 OF THE COMPANIA BENEFICIADORA DE METALES, HACIENDAS DE SAN FRANCISCO, S. A.,

Pachuca, Hidalgo.

The following was supplied by Warner McCann, the general manager for this company. This mill is interesting from the fact that in it were first constructed on the American continent the high tanks invented by F. C. Brown of New Zealand for pneümatic agitation, and as the results so obtained in treating silver ores were better than those formerly obtained by any other system, both in percentage of extraction and cost, as well as in time of treatment, cost of installation, etc., the installation of these tanks has become almost universal in Mexico, and the tanks themselves are known as Pachuca tanks, from having been first installed in this district.

Pachuca tanks.—These tanks, shown in Fig. 19, consist of a steel cylinder ordinarily 44 ft. high and 15 ft. in diameter, terminating below in a cone having an angle of 60° at the vertex. Tanks 60 ft. high and 15 ft. in diameter are being constructed in some mills.

In the lower part of the cone there is a 5-in. pipe with a gate valve for discharging the contents of the tank after treatment. The apparatus in the interior of the tank consists of:

1.—A central tube called the elevator, the diameter of which is 1/10 of the diameter of the tank. This tube is open at both ends, and is suspended in the centre of the tank so that its upper end is 18 in. above the entrance of the pulp into the tank or 20 in. below the top of the tank, while its lower end is 18 in. above the bottom of the cone.

2.—A pipe (a) for compressed air passes through the centre of the elevator and rests on the bottom of the cone. This pipe is closed at the bottom end, but has a number of small holes in it at the level of the bottom of the elevator tube. These holes are covered by a section of rubber hose slipped over the pipe and having its lower end tied to the pipe with wire just below the holes in the pipe, so that the hose forms a sort of collar valve through which the air may escape into the tank, while the pulp is prevented from entering into the pipe. When the pressure of air in the pipe is greater than that due to the column of slime in the tank the air

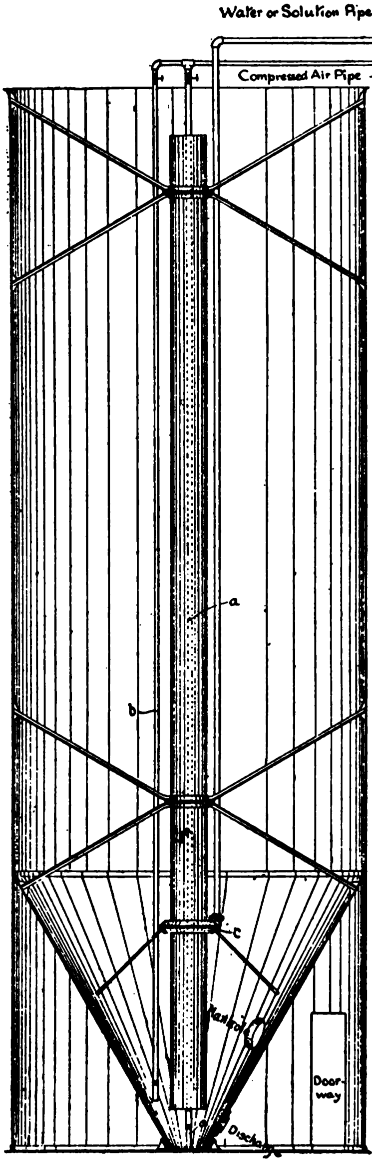


Fig. 19. Sectional Elevation of Pachuca Tank.

escapes through the collar valve and passes up through the elevator tube, carrying the slime with it.

3.—Another air pipe (b) with its collar valve is placed outside of the elevator to keep the slime in circulation during the filling and emptying of the tank.

4. Apparatus (c) consisting of an annular pipe having several small pipes with collar valves screwed into it and so arranged that compressed air, water or solution may be forced through them in order to wash off any sand or slime which may have deposited on the sides of the cone after the agitation has been stopped.

The method of operation to cause the agitation or circulation of the slime in these tanks is very simple, being as follows:

Having filled the tank with pulp and solution, the valve (a) is opened admitting compressed air into the bottom of the elevator where it mixes with the pulp in the elevator, and as this mixture is lighter than the pulp in the tank, it rises to the top of the elevator and overflows into the tank, while another portion of the slime in the tank enters the elevator at the bottom which in turn is raised to the top and overflows, thus obtaining a perfect and continuous circulation of the pulp as long as the compressed air is allowed to enter. Upon starting the circulation the air pressure must be greater than that of the column of slime, but as soon as the circulation is well established less pressure is required, it having been found in practice that while 50 lb. pressure is required to start the circulation, as soon as the sand and slime which have settled in the bottom of the cone have been cleaned out by the scouring of the circulating pulp, the circulation may be maintained by a pressure of but 25 lb. per square inch.

Upon starting the circulation a large quantity of air is required in order to obtain a velocity in the circulating pulp sufficient to remove all of the sand and slime which may have packed in the bottom of the cone, but so soon as this is effected the quantity of air required is much less. The quantity of air required in any particular case depends on the proportion of sand to slime, the fineness of the pulp, and the viscosity of the slime. Ordinarily, in the Pachuca plants, they use 100 cu. ft. per minute, to maintain a tank containing 100 tons of slime in active circulation and to prevent the settlement of the sand on the sides of the cone bottom, but in the Goldfield Cons. Mines Co. in Nevada, and in the plants of F. C. Brown in New Zealand only 30 cu. ft. per minute are used.

Ores treated in the San Francisco mill.—This company has no mines of its own, but purchases the ores from a number of different mines, each of which is different in chemical composition and requires a variation in the treatment.

Milling and cyanidation plant.—During the last year this plant has been entirely renewed, remodeled, and enlarged to a capacity of 300 tons per day, so that, with the exception of the general plan of location of the various departments, which necessarily had to conform with the buildings and walls previously constructed, it is entirely modern and thoroughly up to date.

Crushing and sampling.—The plant contains a complete sampling mill where all ores are received, crushed, automatically sampled, and dumped into bins, from which they are carried to the battery-bins in cars.

Milling, concentration and classification.—The batteries consist of 40 stamps, weighing 1250 lb. each, which drop 7½ in. at the rate of 100 drops per minute. The ore is crushed in the batteries to pass through a No. 9 roll slot screen and thence falls on to 8 Wilfley concentrating tables, whence the tailing runs into 4 Dorr classifiers, from which the slime and excess solution fall into 4 Frenier pumps, by which they are elevated to 2 Dorr slime-thickeners, 24 ft. in diameter by 10 ft. deep. The sand from the Dorr classifiers falls into 4 Krupp tube-mills, 4½ ft. in diameter by 13 ft. long, so placed that each mill receives the sand from one classifier. After the sand is reground in the tube-mills, it passes into 4 Frenier pumps, which return it to the Dorr classifiers until the whole of the pulp has been converted into slime which passes to the 2 Dorr slime-thickeners previously mentioned.

From the bottom of these slime-thickeners the slime of the proper consistency for good concentration is drawn off and fed to 11 Johnston concentrators and 4 Deister slime tables, by means of which the heavy minerals which have been liberated by regrinding, or which escaped concentration on the Wilfleys, are concentrated out of the slime. The tailing from the Johnston and Deister tables are elevated by a 3-in. centrifugal pump to a third Dorr slime-thickener 24 ft. in diameter by 10 ft. deep, whence the thickened slime, containing 1 ton of dry slime to 1½ of solution, flows to the Pachuca tanks for treatment, while the overflow of this as well as the overflow of the other 2 slime-thickeners flows to a tank from which, when clear, it is pumped to the tanks above the mill which supply solution to the batteries. It sometimes occurs, when milling certain classes of ore, that the third Dorr slime-thickener will not have a sufficient capacity for settlement of the ore milled, so that the overflow contains unsettled slime. When this is the case the overflow from this thickener is run into a series of four masonry settling-tanks, where the slime is settled before pumping the solution to the tanks which supply the batteries.

Cyanidation.—There are 8 Pachuca tanks in this plant, each of which is charged with from 80 to 100 tons of dry slime for treatment. The slime resulting from the milling and classification,

which is treated in these tanks, is of such a fineness that 80% will pass through a 200-mesh screen. Samples are taken every 12 hours during the treatment to determine the gold, silver, protective alkali and total active cyanide contained in the solutions, as well as the gold and silver contained in the dry slime, after washing the slime sample with water to remove any solution containing values which might vitiate the assay of the slime. The results of these assays determine the time of treatment required by each ore.

Filtration.—As soon as a satisfactory extraction has been obtained in the Pachuca tanks the slime contained therein is discharged through the discharge pipes in the bottom of the cone which leads to a centrifugal pump by which the slime is elevated to a slime storage tank located above the filters, from which it is fed to the filters as required. There are two filter-plants in this

Fig. 20. Butters Filter at San Francisco Mill.

mill, each having a capacity of 150 tons per day. The Butters filter has 78 filter-leaves, while the Moore filter has two baskets of 40 leaves each. Each filter-plant is supplied with all of the latest improvements and they give entire satisfaction. After filtration the solutions are pumped to the sand filters above the precipitation room, while the slime tailing, after being filtered and washed, is discharged into the tailing-dam. The solution from the tailing-dam, after settlement of the slime, is pumped to a tank above the filters for use in them as wash water.

Precipitation.—The precipitation room contains 10 zinc-boxes made of sheet steel, each 15 ft. long, 3 ft. wide, and 2½ ft. deep, which are divided into 5 compartments each. The precipitation of the clarified solutions from the sand filters is almost perfect in these boxes, as the solutions entering the boxes assay from 100 to 250

gm. of silver and from 1 to 3 gm. of gold per ton, while the precipitated solutions leaving the boxes do not carry more than 1 gm. of silver and 0.1 gm. of gold per ton.

Clean-up and melting.—The details of the practice in this mill in cleaning up and melting the precipitate is hereafter described in Chapter XVIII.

Extraction and costs.—As this is a custom mill, the details of extractions and costs are not available for publication.

CHAPTER XI.

CYANIDE PRACTICE OF THE SAN RAFAEL Y ANEXAS CO.,

Pachuca, Hidalgo, Mexico.

The following description is an extract of an article written on this subject for the Mexican Institute of Mining and Metallurgy, on November 26, 1909, by E. Girault, general manager of the company.

This mill, built in 1908, belongs to the type introduced the year before in the San Francisco Hacienda.

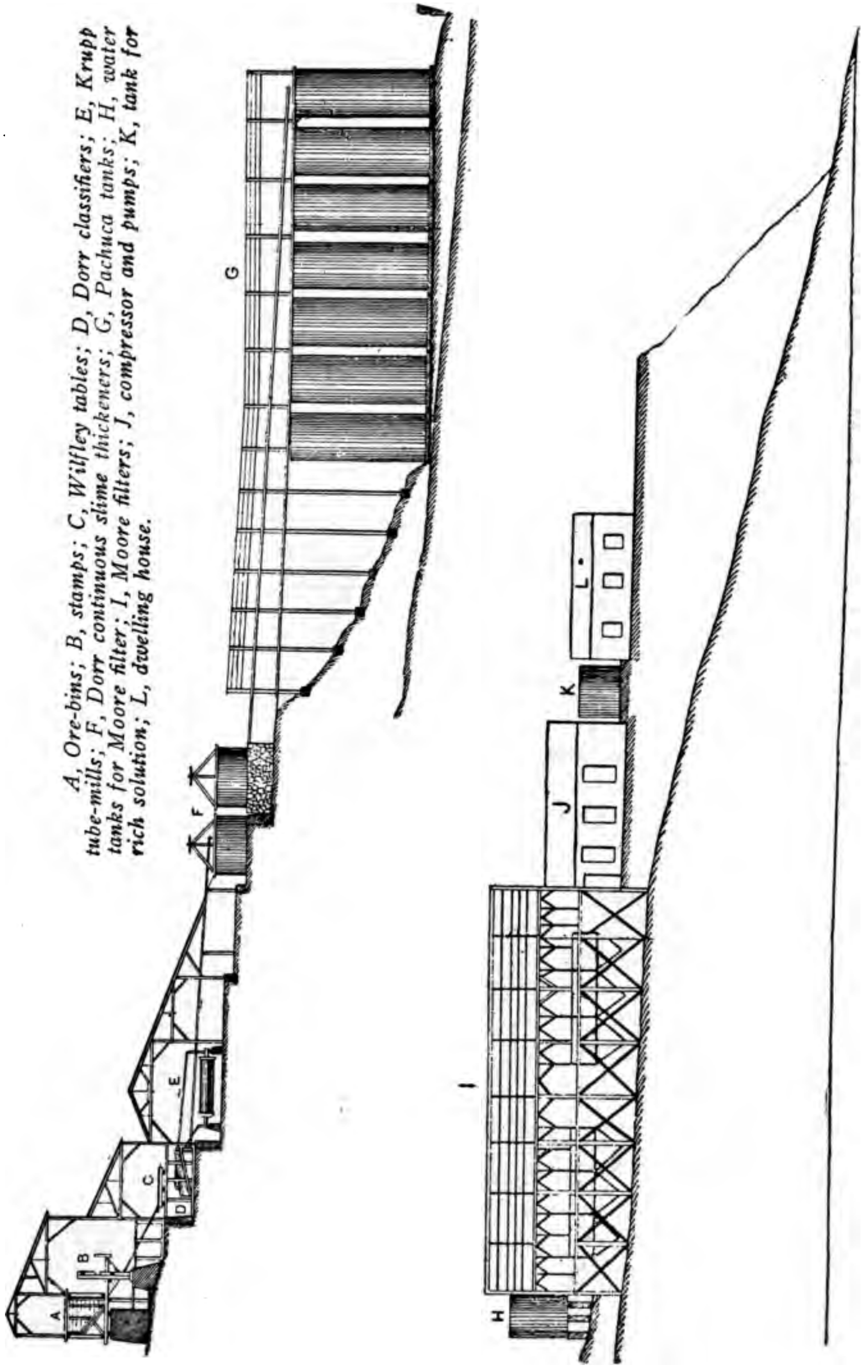
Ores.—The ores from the San Rafael mines, derived from the Vizcaina vein (the mother lode of the district), contain from 70 to 75% of silica, and from 10 to 20% of calcite. The silver, mixed with a small amount of base sulphides, is found mostly in the State of Ag_2S. The gold occurs in a nearly constant proportion of 4 to 5 gm. per kg. of silver in the low, and of 3 to 4 in the high-grade ores. All ores above 300 gm. of silver to the ton are cyanided.

Treatment.—The ore is hand picked; crushed in Blake crushers and sampled in the mine yard; ground in 0.25% KCN cyanide solution in stamp-batteries; concentrated on Wilfley tables; classified in Dorr classifiers; reground in Krupp tube-mills; settled in Dorr pulp-thickeners and agitated 'Pachuca' tanks, which were invented in New Zealand in 1904 by Brown and McMicken. The solutions are filtered in Moore vacuum-filters and precipitated on zinc-shavings. Fig. 21 and Figs. 22 and 23, show the plan and section of the mill.

Power and water supply.—The mill is driven by electric power. The motors are of the Westinghouse C. C. L. type. Each battery of 20 stamps has a 75-hp. motor; each Krupp tube-mill one of 100 hp. The total power consumption is 1.68 hp. per day, per ton of ore. The water supply is drawn from the mine; the water is slightly alkaline.

Milling.—There are 80 stamps in use; 40 Krupp, weighing 850 lb., set on timber foundations, and 40 Allis Chalmers of 1250 lb., bolted to concrete foundations. The light stamps drop 7¾ in. and the heavy ones 6¾ in. 104 times per minute. The shoes and dies of forged steel last from 90 to 100 days.

Tyler's ton-cap screens, Nos. 58 and 365, equivalent to 10-mesh, 18 wire, 1.32 mm. opening, and to 12-mesh, 20 wire, 1.067 mm., re-



A, Ore-bins; B, stamps; C, Wilfley tables; D, Dorr classifiers; E, Krupp tube-mills; F, Dorr continuous slime thickeners; G, Pachuca tanks; H, water tanks for Moore filter; I, Moore filters; J, compressor and pumps; K, tank for rich solution; L, dwelling house.

Fig. 21. Profile of Cyanide Mill of the San Rafael y Anexas Co., Pachuca.

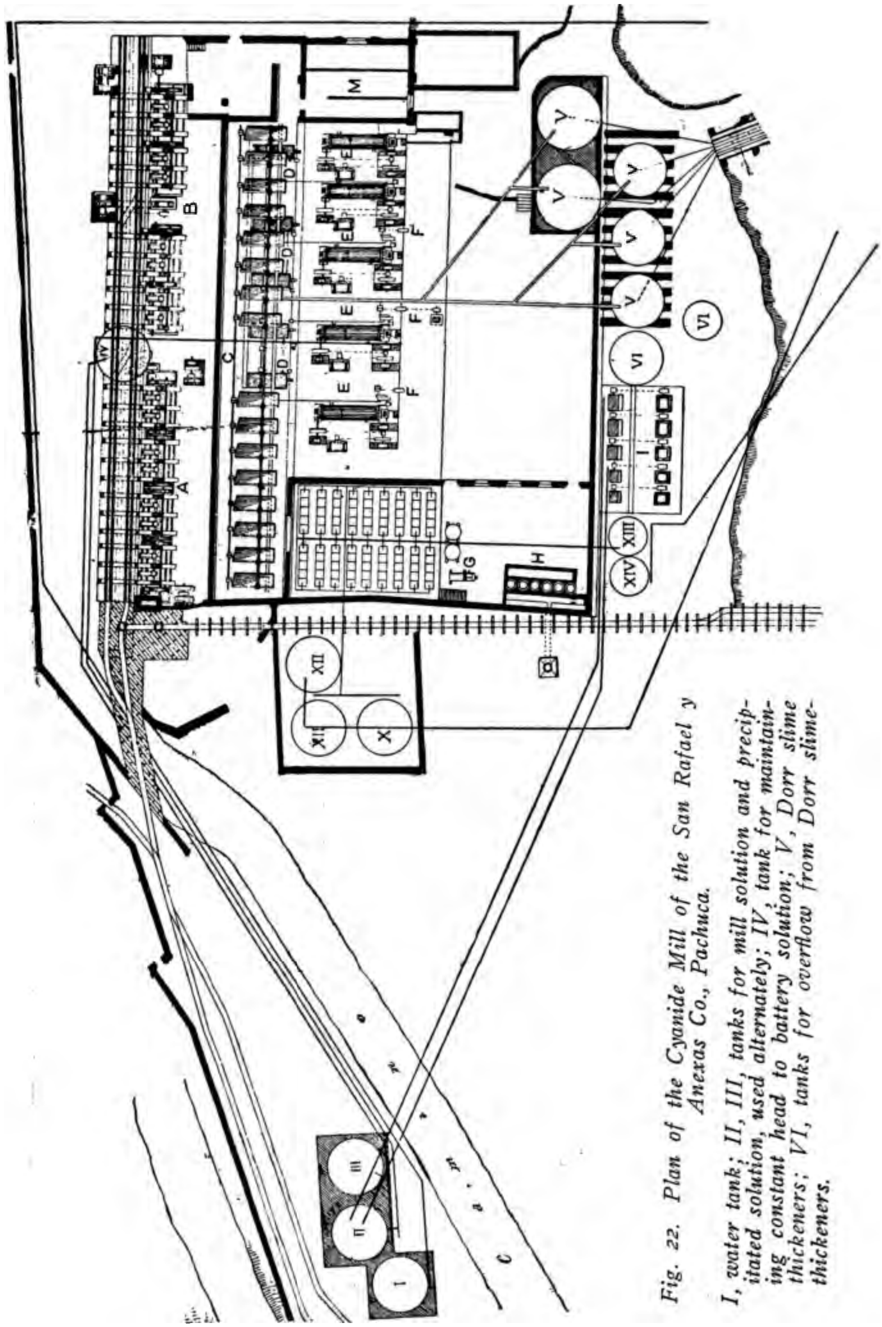


Fig. 22. Plan of the Cyanide Mill of the San Rafael y Anexas Co., Pachuca.

I, water tank; II, III, tanks for mill solution and precipitated solution, used alternately; IV, tank for maintaining constant head to battery solution; V, Dorr slime thickeners; VI, tanks for overflow from Dorr slime-thickeners.

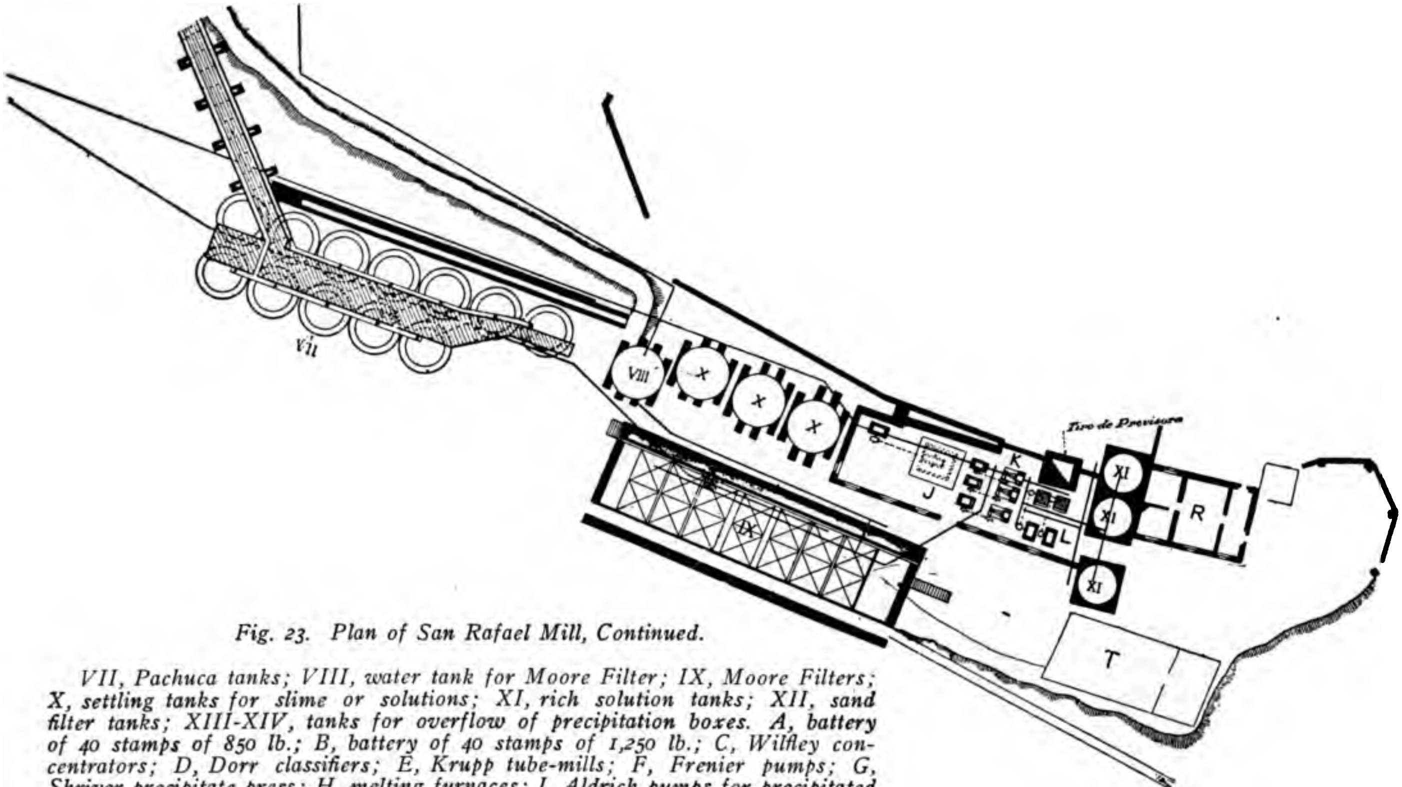


Fig. 23. Plan of San Rafael Mill, Continued.

VII, Pachuca tanks; VIII, water tank for Moore Filter; IX, Moore Filters; X, settling tanks for slime or solutions; XI, rich solution tanks; XII, sand filter tanks; XIII-XIV, tanks for overflow of precipitation boxes. A, battery of 40 stamps of 850 lb.; B, battery of 40 stamps of 1,250 lb.; C, Wilfley concentrators; D, Dorr classifiers; E, Krupp tube-mills; F, Frenier pumps; G, Shriver precipitate press; H, melting furnaces; I, Aldrich pumps for precipitated and mill solutions; J, compressor; K, vacuum pump; L, rich solution pumps; M, electric sub-station; N, blacksmith shop; O, machine shop; P, store-room; Q, electrical machinery repair room; R, dwelling houses; S, carpenter shop; T, filter repair room.

spectively, are used in the batteries. These screens last from 30 to 40 days. The 850-lb. stamps crush, through 10-mesh, from 3 to 3½ tons per day; the 1,250-lb. from 6 to 6½ tons through 10-mesh; and from 5½ to 6 tons through 12-mesh. From 7 to 8 tons of solution to 1 of ore is used in the batteries. One pair of spring rolls 36 by 16 in. and two more tube-mills have been ordered; when installed 8-mesh screens will be used in the batteries, expecting to raise the crushing to about 500 tons per day.

The following is a sizing test of the battery pulp; No. 58 ton-cap.=10-mesh screens being used:

Screen	+40	+60	+100	+150	+200	−200
Per cent.	37	9	7	13	4	30

The total power consumption in grinding, including crushers, stamps and tube-mills, is 1.05 hp., day per ton. The mill solution is stored in two tanks, II and III, of 200 cubic meters each; these tanks are used alternately to supply the mill, and to receive the barren solution, being worked in cycles in order to avoid the enrichment of the solution. According to the class of ore treated from 6 to 12 kg. of lime are added in the bins per ton of ore. The alkalinity of the solutions is kept at 1 kg. of CaO to the ton. The mill solution averages 0.25% of KCN; the cyanide loss in crushing is about 600 gr. to the ton of ore. The extraction in the grinding process is high, averaging during October 38.3% of the silver and 70% of the gold contained in the ore.

Concentration.—Sixteen Wilfley tables, one for each five-stamp battery, are used. From 17 to 30 tons of ore per day are fed to each Wilfley without overloading them. The Wilfley tables are also used for unwatering the pulp and as auxiliary classifiers; the slime overflow being sent to pulp thickeners.

During the month of October 22.09% of the values were recovered in concentrates containing:

Silver	25.586 kg. per ton.
Gold	126 gm. per ton.
Iron	34.45 %
Silica	12.25 %
Zinc	3.40 %
Sulphur	40.50

The concentrate is sold to the smelters.

Classification.—Five Dorr classifiers, one for each tube-mill, are used. From 50 to 70 tons of pulp, and all the return from the tube-mill are fed to each classifier; the pulp being in the proportion of 1 to 2.38 of solution; the discharge contains 1 of solution to 2 of dry sand. The Dorr classifiers have been working continuously for

one year without giving any trouble. The classified slime discharge averaged in October:

+100	2.5%
+150	15.5 "
+200	7.5 "
—200	74.5 "

Tube-mills.—Five Krupp No. 5 tube-mills, 4 by 20 ft., with Neal's baffle at both ends. Each of these mills is equipped with a 100 hp. motor, but it has been determined that 75 hp. motors would be sufficient. These mills were first equipped with smooth liners of hardened Krupp steel, but as they lasted but 90 days, the El Oro liner of white iron was substituted. These lasted for 5 months before being worn to such an extent that they would not retain one-half of the pebbles in the grooves. They were then replaced with El Oro liners of Krupp steel, which lasted 8 months and are still as good as new. Both Danish and French pebbles have been used with good results, charging 2 sacks of pebbles four or five times a week, through the Neal's baffles. The experiment of using mine quartz instead of pebbles was unsatisfactory. All of the discharge from the tube-mills is returned to the Dorr classifiers by Frenier pumps. The 5 Frenier pumps are run by one 10 hp. motor and during the year have given perfect satisfaction. The capacity of each tube-mill is rated at 50 tons of sand per day; the heads averaging:

+100	48.0%
+150	39.6 "
+200	4.2 "
—200	8.2 "

The heading, which comprises the sand discharged from the Dorr classifiers, contains but 60% of moisture, and in order to secure the proper results in regrinding, it is diluted upon entering the tube-mill, with sufficient solution to bring up the proportion to 1 of slime to 1.5 of solution.

Pulp thickeners.—Five Dorr pulp-thickeners, receiving slime diluted 1 to 10 and discharging 1 of dry slime to 1.2 of solution, are employed to unwater the pulp previous to agitation in the Pachuca tanks. Three of the thickeners, 20 by 10 ft., are supplied with 70 tons dry of pulp each; the other two, 24 by 10 ft., will take 100 tons per day, and all overflow clear solution.

Pachuca tanks.—Ten 15 by 45-ft. Pachuca tanks are used for the agitation of the pulp at present; two more have been ordered and are soon expected. The average charge is 100 tons of dry slime and 120 tons of solution. One to one of solution (120 tons of slime to 120 of solution) has been tried; as well as 1 to 2, and 1 to 3 of solution; but the best proportion was found to be 1 of ore to 1.2 of solution.

Compressor.—One Rand-Ingersoll air compressor 18¼ in. diam. by 16 in. stroke, delivering at 100 rev. 700 cu. ft. of air at 9000 ft. elevation, is employed for supplying air to the Pachuca tanks, to the Moore filter-tanks, and others. The amount of air required for an energetic agitation in the Pachuca tanks is estimated in San Rafael at 100 cu. ft. per minute, at 20 lb. pressure.

Agitation.—Before commencing the agitation in the Pachuca tanks, enough cyanide is added to bring the solution up to 0.3% KCN, and also 250 gm. of lead acetate is added per ton of ore contained in the tank. The pulp is agitated during 36 hours, left to rest from 12 to 24 hours, as it has been found that during this period the silver extraction increases about 20 gm. The solution

Fig. 24. Exterior of Pachuca Tanks.

in the Pachuca tanks after agitation contains from 400 to 450 gm. of silver and 0.33 gm. of gold per ton.

Filtration.—Two units of the type A No. 3 of the Moore filter are employed; a third unit has been ordered. The leaves are 10 by 6 ft. and their capacity is 2.5 tons of dry slime per day; 160 leaves in four baskets of 40 each are used; and 80 more are kept in reserve for repairs. Two traveling cranes, worm geared, with 30 hp. variable speed G. E. motors for lifting, and 5 hp. for traveling, are used for transferring the baskets. According to the condition of the leaves a one-inch cake is formed in from 20 to 30 minutes. One hour barren solution wash, and 15 minutes water wash are given to the cakes. On each basket, one 16-in. by 20-ft. spiral pipe, serves as vacuum chamber, and holds the cake during the 5 minutes

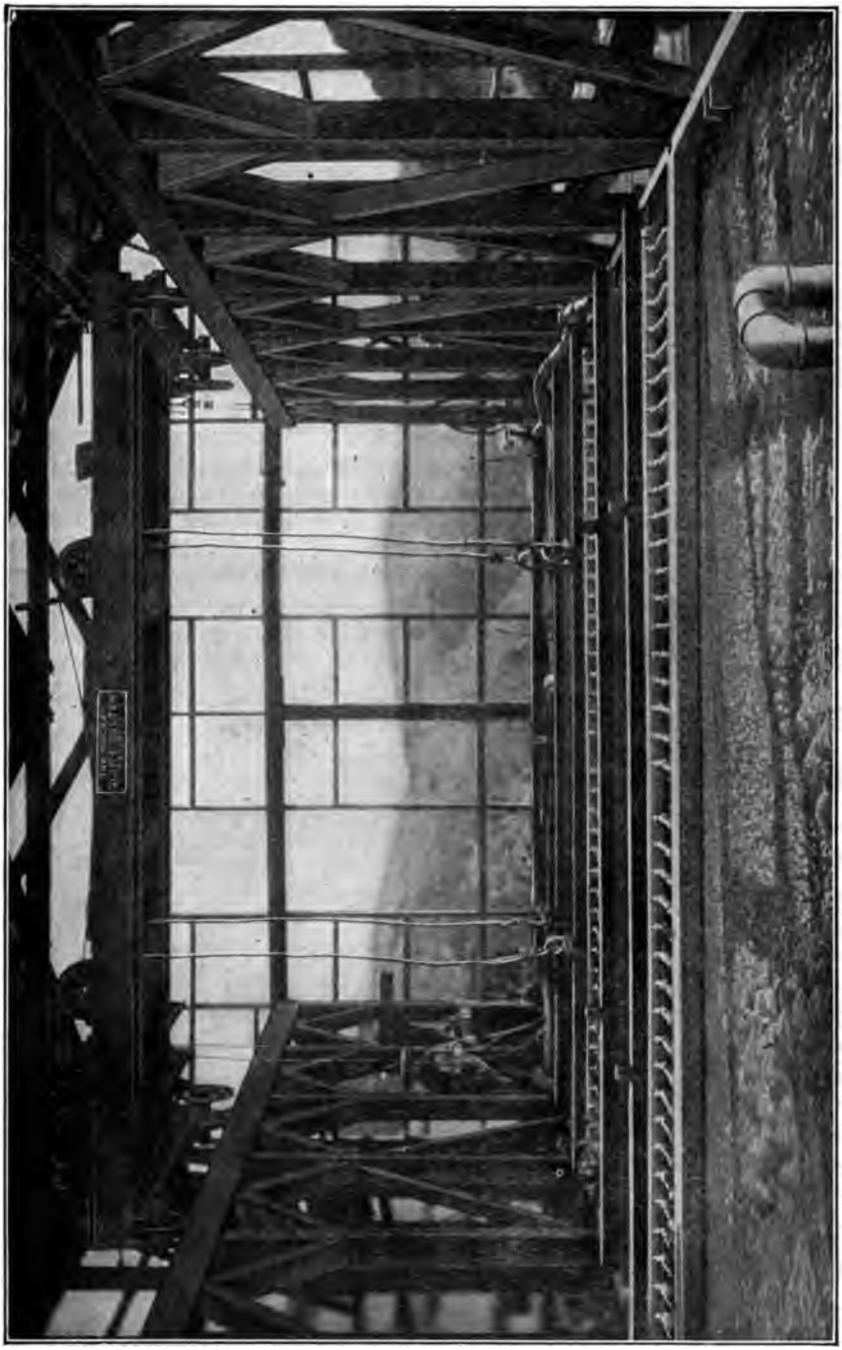


Fig. 25. Moore Filter at San Rafael Mill, Pachuca.

of the transfer. In order to clarify the solutions, sand filters are used before the zinc-boxes. During October the water discharge assayed 4.5 gm. of silver to the ton; the cake 55 gm. unwashed, and 50 gm. washed. During November, treating higher grade ores, and employing three baskets only, one of them in bad state, the cake discharged assayed 71 gm. of silver unwashed, and 63 gm. washed.

Precipitation.—There are 20 precipitation-boxes of 5 compartments each 3 by 3 by 2½ ft. but as only 4 compartments in each box are used for precipitation, this gives a total capacity of 1120 cu. ft., containing 700 cu. ft. of zinc-shavings, or 1 cu. ft. of zinc-shavings per ton of solution to be precipitated. Zinc-shavings, 0.006 inch thick, are used in precipitation. Two tons of solution to the ton of ore are precipitated. The heading in the boxes assays from 200 to 300 gm. of silver; the tailing 2 gm. The clean-up is made weekly. The short-zinc which remains on a 20-mesh screen is returned to the boxes where they are placed in layers 4 in. thick, alternated with layers of new zinc-shavings. Lumps of cyanide are placed in these boxes to hasten their solution; the shorts on 40-mesh are melted separately. They contain 75% of silver and 10% of zinc. The precipitates are pressed and dried to 20% moisture on a 24 by 24-in. Shriver press, through which they are forced by a 4 by 4-in. Gould triplex pump.

Melting.—Coke furnaces and Dixon's No. 300 graphite crucibles are employed. The precipitate is melted with the following flux:

	Per cent.
Broken glass	15
Borax-glass	6
Sodium carbonate	4

The short-zinc, which remains on the 40-mesh screen, is fused with:

	Per cent.
Broken glass	20
Borax	8
Sodium carbonate	6

The bars, without remelting, assay from 920 to 950 fine in silver and 5 fine in gold. The bars from the short-zinc assay from 700 to 800 fine in silver. The slags and ashes are crushed quarterly in a battery and concentrated on a Wilfley table. These slags, of which one ton is produced from the melting of 40 or 50 bars of bullion, assay about 18 kg. of silver per ton. Of this amount 90% is extracted in concentrates, which are melted into bars. The tailing from the concentration of the slag, which assay from 1.800 to 2 kg. of silver per ton, were formerly treated in the cyanide plant, but latterly, as it was found that the coke and sulphur present in the coke, prevented a high extraction, they have been sold to the ore-buyers.

RÉSUMÉ.

The résumé of the November results is given below: Tons crushed, 8,393.

Assays

Ore contents:	
Silver	0.901 kg.
Gold	4.43 gm.
Concentrates:	
Silver	25.348 kg.
Gold	141.6 gm.
Heads of Pachuca tanks:	
Silver	0.397 kg.
Filter discharge:	
Unwashed, silver	0.071 kg.
Washed, silver	0.063 kg.

Extraction:

In concentration, silver	24.19
Cyaniding:	
Crushing, silver	31.74
agitation, silver	36.18
Total extraction, by assays, silver	92.11
Total extraction, by bullion, silver	91.43
Total extraction, by bullion, gold	97.08

Consumption per ton of ore:	Kg.
Sodium cyanide, 128 p%	1.078
Lead acetate	0.317
Lime	6.706
Flint pebbles	1.227
Zinc	0.843
Coke	1.646
Borax	0.084
Sodium carbonate	0.044

The cost per ton was:

General expenses	₱0.78
Grinding and concentrating	1.18
Cyaniding	1.32
Pumps and compressor	0.15
Filtration, including royalty	0.33
Precipitation and melting	0.44
Total	₱4.20

CHAPTER XII.

CYANIDE PRACTICE AT THE PLANT OF THE LUCKY TIGER-COMBINATION G. M. CO.

At El Tigre, Sonora.

By D. L. H. Forbes.

The mine of the Lucky Tiger Company is situated in the Moctezuma mining district of Sonora, 60 miles to the southeast of Douglas, Arizona. The ore is extracted from two veins known as the El Tigre and the Sooy, the first of which has been developed for a distance of 4000 ft. by adits and tunnels, the levels being spaced approximately 100 ft. apart. The lowest workings are on the No. 7 level, where an adit 2400 ft. in length was driven to cut the El Tigre vein at a depth of 850 ft. below the "A" level, and to provide an easy outlet for all ore above it. This adit has its entrance just above the concentrators and a surface tramway extends from it over a 250-ton steel ore-bin at the breaker plant. The property is situated in a region of such precipitous mountains that railroad facilities are out of the question and high-grade ore, concentrate, and cyanide bullion have to be hauled a distance of 30 miles to the Nacozari railroad by mule-back or wagons.

The ore in the El Tigre vein below the No. 3 level consists of sulphides of lead, copper, zinc, iron, and silver in a quartz gangue. Gold is present to the extent of about 0.006 of the silver by weight. The vein is seldom more than six feet in width and considerable quantities of andesite and talcose material from the walls becomes mixed with the ore in the course of mining. The ore in the Sooy vein contains a larger amount of the sulphides of copper, lead, and zinc for the same assay in silver that is carried by ore of the El Tigre vein, but its gold content is relatively higher. Both veins have extraordinary richness in spots, so that sorting of high-grade ore has to be done both at the mine and at the mill. Besides the high-grade ore that is shipped direct to the smelter, the mine produces from 150 to 180 tons per day of ore carrying 30 oz. silver and 0.15 oz. gold per ton of 2000 lb. This ore is sent to the milling plant. The cyanide plant treats the tailings from the two concentrating mills and also 50 to 75 tons per day of tailing from old dumps that have accumulated below the mills.

The run-of-mine ore, stored at the breaker plant, gravitates over an automatic feeder and shaking grizzly to a gyratory crusher, where the coarse lumps are broken and spread on a Robins picking

belt. The breaking and sorting are done only on the day shift and the capacity of this section of the plant is 20 tons per hour. After being automatically weighed and sampled, the broken ore is distributed between No. 1 and No. 2 mills by means of belt conveyors and automatic machinery.

No. 1 mill is the old concentrator with a few slight modifications and additions. Formerly the mill was driven by a 150-hp. Corliss steam-engine, and with electric power for lights and auxiliary motors furnished by two Weber gas-engines and suction gas-producers. Much cheaper power is now obtained from Douglas over a 65-mile electric transmission line and by using individual motor drives wherever possible in the milling plant. The capacity of No. 1 mill is 90 to 100 tons per day and the power consumption for motors and lights averages 167 hp. The scheme of treatment consists of roll crushing, screening in trommels, concentration on Hartz jigs and Wilfley tables, regrinding in 5-ft. Huntington mills, and vanner concentration of the slime.

The No. 2 mill, on the other hand, for the same capacity consumes only 86 hp. for motors and lights. The treatment consists simply of crushing through 20-mesh screens with twenty 1200-lb. stamps, hydraulic classification, concentration of the sands on Wilfley tables, concentration of the slime on Deister tables, regrinding of the Wilfley middlings in a 5-ft. grinding pan and treatment of the product along with the middlings of the Deister tables on a Deister retreatment slime table. No elevators are used in No. 2 mill. One American battery man with two Mexican table attendants constitutes the operating force on each shift. For November, 1911, the milling cost per ton was ₱1.34, as compared with ₱3.79 in No. 1 mill, and, while the milling cost for No. 1 mill will undoubtedly be lowered in the future, it is safe to state that the cost of stamp-mill concentration in No. 2 mill is less than one-half the cost of stage-crushing and concentration in No. 1.

Comparisons of results in Mills Nos. 1 and 2, November, 1911:

	No. 1 mill, stage crushing and concentration.		No. 2 mill, stamp milling and concentrating.	
Total milling cost per ton..		₱3.791		₱1.346
Crushing—				
General labor..........	₱0.498		₱0.453	
Power.................	0.383		0.279	
Repair parts...........	.....		.....	
Superintendence........	0.065		0.047	
Miscellaneous supplies..	0.046		0.048	
		₱0.992		₱0.827
Regrinding—				
General labor..........	₱0.371		₱0.024	
Power.................	0.055		0.016	
Repair parts...........	0.010		.....	
Superintendence........	0.062		0.002	
Miscellaneous supplies..	0.040		.....	
		0.538		0.042

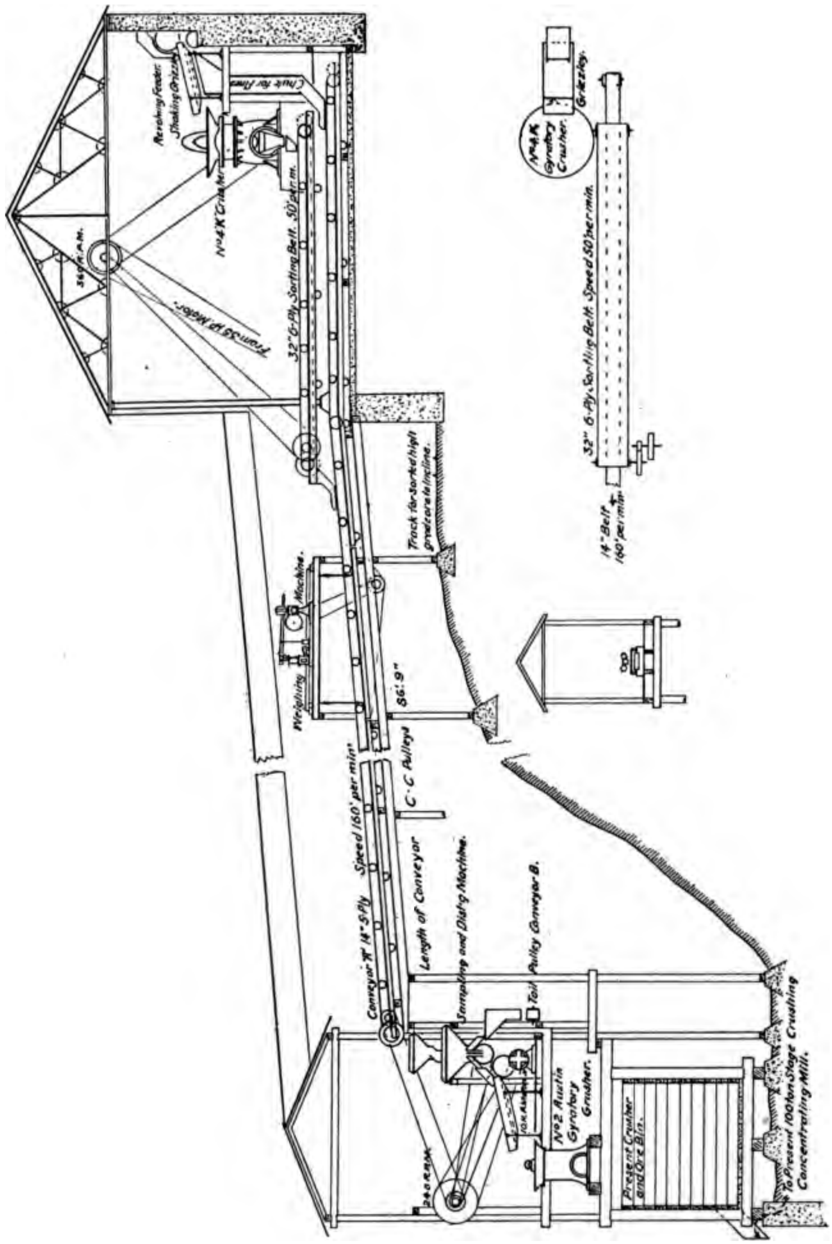


Fig. 26. Coarse-Crushing Plant at El Tigre.

	No. 1 mill, stage crushing and concentration.		No. 2 mill, stamp milling and concentrating.	
Concentration—				
General labor	₱1.022		₱0.263	
Power	0.491		0.096	
Repair parts	0.057		0.001	
Superintendence	0.096		0.040	
Sampling and assaying	0.096		0.040	
Miscellaneous supplies	0.178		0.022	
		₱1.922		₱0.477
Construction		0.339		....
Recovery of precious metals—		Per cent		Per cent
Gold		68.6		73.8
Silver		58.3		60.4
Value of both		59.6		62.6
Ratio of concentration		16.6 into 1		18.5 into 1
Average assay in Ag, ore milled		28.08 oz.		31.29 oz.
Average assay in Ag, concentrate		270.86 oz.		349.99 oz.
Average assay in Ag, tailings		12.48 oz.		13.08 oz.

On account of the large quantity of sulphides present in the ore which decompose cyanide, concentration has to be performed in water, sufficient lime being added to the ore at the breaker plant to make the mill water alkaline and to aid in the settlement of slime.

Cyanide treatment of the tailing consists of fine grinding of the sand in tube-mills, dewatering of the slime by Dorr thickeners and vacuum-filters, treatment of reground sand and slime in Pachuca tanks, pressure filtration in Kelly filters, and precipitation of solutions by the Merrill zinc-dust process.

There are five 5-ft. by 14-ft. tube-mills for fine grinding. These are all equipped with scoop feed ends and spiral worm discharge ends for the charging of pebbles. The Forbes type of tube-mill liners have been found to give better service than the El Oro liners. The former consist merely of grating plates of chilled iron or manganese steel which are backed by mild steel plates 3/16 in. in thickness. In this way the advantages of a lining in which the pebbles wedge are retained while the backing pieces do not have to be thrown away when the ribs are worn down, with the result that there is a better efficiency in the wear of the iron. Both mine quartz and Danish flint pebbles have been tried, with results in favor of the latter. With Danish pebbles the consumption of grinders is only about one-tenth of that when quartz is used, while the wear on linings is less and a better product is given; so that, if quartz is used at all, only a small proportion is added to the charge and then only when the supply of flints happens to be running low.

The treatment tanks are slightly modified from the standard Pachuca pattern and are arranged in two rows of four tanks for the continuous treatment system. Each tank is 15 ft. in diameter by 40 ft. in height and the central tube rises only two-thirds of the

distance from the bottom. No check valve of any kind is used on the end of the compressed-air pipe. The pipe descends vertically through the central 18-in. tube and stops just flush with its bottom. In this way a vigorous submerged agitation is maintained in the tanks and the troublesome rubber check valves have been eliminated. In no case has the air pipe become choked with sand or slime plugs at its mouth. An emergency hose and line of pipe is kept ready for attachment to a high-pressure air line over the tanks and may be used either inside or outside the central tube for starting agitation after a shut-down of power. All tanks are arranged also for a withdrawal of the pulp from the bottom and the elevation of the same with a centrifugal pump in case of emergency. The connections for the continuous system are so arranged that they may be blown out whenever necessary with high-pressure air, which dislodges any material that may have settled in the pipes during times when the draw-off at the eighth tank is throttled and the flow between the tanks checked. By means of a system of three-way cocks, the flow from any particular tank may be diverted for a short time to any of the others beyond its neighbor, thus giving the system great flexibility and overcoming many of the difficulties experienced elsewhere in the operation of the continuous Pachuca tank treatment. The flow at full capacity through the eight tanks gives a length of treatment of about 48 hours. The solution in the first tank of the series is made up to a strength of 0.3% KCN by allowing the incoming pulp to fall on and dissolve cakes of sodium cyanide in a perforated iron basket. The protective alkalinity is maintained at from 1 to 1.5 lb. CaO per ton of solution. The dilution of the pulp in the treatment tanks is 1 of solids to 2 of liquids by weight. No classification has been observed to take place in the tanks, the screen tests of samples from the last tanks being practically the same as those from the first.

After treatment the pulp is thickened to a consistency of 1 to 1.5 in two steel slime thickeners of the Tigre type and then is stored in a 30-ft. by 12-ft. mechanically agitated tank above the filters.

The filtration plant consists of five Parral type Kelly presses. The cylindrical shell of each press is 5 ft. in diameter by 14 ft. in length and is set with the long axis at an inclination of 9 deg. 30 min. from the horizontal. Each unit has 13 leaves with a total filter area of 1200 sq. ft. The capacity per unit on the El Tigre slime is about 50 dry tons per 24 hours, or 0.04 tons per square foot filter area per day. The filtration is done by pressure from the 30-ft. by 12-ft. slime storage tank, which is placed 60 ft. higher than the presses. The excess slime after the formation of cakes on the leaves is allowed to gravitate to a tank below the filters, from which a pair of Frenier sand pumps in tandem elevates it again to a mechanically agitated storage tank set just behind the presses at

an elevation convenient for filling. In this way the presses are rapidly filled with slime through a 12-in. pipe-line at low pressure and then high-pressure slime from the upper tank is fed for the filtration and building of cakes on the leaves. All five filters are operated from a central switchboard as one unit. The operations of filling and forming cakes takes about 24 minutes, with the filtering pressure varying from 10 to 32 lb. per sq. in. The cake is built up to a thickness of from ⅞ in. to 1⅛ in. Discharging excess slime and filling with barren solution takes 6 minutes. The barren wash is given for a period of about 30 minutes and at a pressure of 25 to 34 lb. The water wash and drying of the cakes with compressed air take about 15 minutes. Discharging the cakes is very trouble-

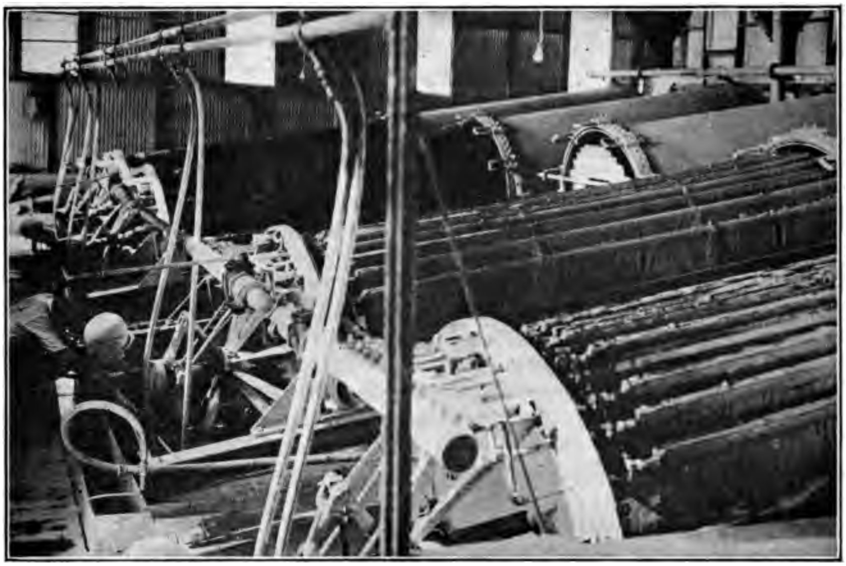


Fig. 27. Kelly Filters at El Tigre.

some, owing to the sticky nature of the slime, and it usually requires about one hour from the time the carriage is run out until all the leaves are cleaned and the carriage run back ready for a fresh charge. The cakes, after air-drying on the leaves, contain only 28% moisture, but this amount is increased to nearly 48% by the time the cakes are disintegrated and washed out into the discharge launder. As water is very scarce at certain seasons of the year, the Kelly filter was chosen principally on account of the fact that, even with sticky clay slime, it may be made to operate with a small quantity of water for discharging the residues. The cost of filtration is high, however, being at present around 60 centavos per dry ton of slime treated, and it is hardly expected, even after the

operators have acquired more skill, that the total cost of labor, power, chemicals and supplies will be less than 40 centavos per ton. The loss of dissolved gold and silver discharged with the residues as shown by the difference in the assays of washed and unwashed samples, varies from 4 to 20 centavos per ton, with the average around 14 centavos. Considering the fact that the solution carries ₱5 per ton in the value of the gold and silver, this filtration loss is not excessive.

Only about 30% of the bulk of the solutions coming from the Kelly filters requires to be clarified. The balance, which is the effluent after cakes over ¼ in. in thickness have formed, is perfectly clear and needs no further treatment previous to precipitation. The cloudy solutions from the filter plant and the solution overflow from the Tigre thickeners are clarified in locally designed gravity filter tanks that consist of vertical filter leaves spaced 18 in. apart at the bottoms of tanks 10 ft. in height. These leaves have no stitching and have been found to be much better than the old sand filter bottoms which they replaced.

The precipitation equipment consists of two Merrill triangular frame 52-in. presses, one with 20 frames and the other with 30 frames, 3 in. deep. About 500 tons of strong solution and 700 tons of weak solution are precipitated daily. Zinc-dust is fed to the solutions by means of two of the improved Merrill hopper-and-worm type of feeders. The addition of a small amount of powdered lead acetate to the zinc-dust in the hoppers improves the precipitation and raises the grade of the bullion. On account of the large amount of copper in the solutions, it is very doubtful if the old zinc-shaving method of precipitation would have worked at all, while the zinc-dust process, on the other hand, has been entirely satisfactory from the start and requires a minimum of attention from the operators. The ratio of zinc used to fine gold and silver produced is 1.52 to 1. The strong-solution press is cleaned up once a week and the weak press bi-monthly. No. 10 standard cotton duck, with sheet covers of cotton sheeting, is used for the filtering medium in the presses. The covers are changed for ones that have been washed at every clean-up and last for about three campaigns, when they are again washed, dried, and burned, the ashes being melted with precipitate. By exercising care in the clarification of all solutions that are to be precipitated, the pressure in the presses will not rise above 15 lb. before the press is two-thirds full of precipitate. The strong solution precipitate carries from 62 to 65% of silver and about 10% of copper, while the weak solution precipitate runs 40 to 45% silver and about 21% copper. During precipitation the strong solution gains in strength. Previous to precipitation it is 0.19% KCN, while after precipitation it titrates 0.22%, a gain of 0.03%. The same phenomenon is observed to a lesser degree in the case of the weak solution.

Melting is done in two No. 125 tilting Steele-Harvey furnaces using gas oil for fuel. No acid treatment of the precipitate is necessary and it is simply mixed with flux and melted in the ordinary manner. As a large part of the copper becomes oxidized in the course of drying the precipitate, enough silica is added in the flux to carry off both copper and zinc oxides in the slag. Bullion bars 850 to 920 in fineness are produced at one melting.

At first sight it would appear that El Tigre concentrate could be profitably treated by cyaniding at the mill, and elaborate experiments were conducted to determine this point. It was found in treating the regular concentrate from the mill that carried 3 oz. gold, 365 oz. silver, and 1.6% copper, by any of the ordinary methods resulted in enormous consumption of cyanide and but a poor recovery of the precious metals. It was noticed, too, that copper went into solution ahead of either gold or silver during the first 40 hours of treatment by successive air agitations with fresh solutions of cyanide. Preliminary treatment with sulphuric acid showed no appreciable benefit. Finally, however, a method was evolved that seemed to offer possibilities. By treating the concentrate after extremely fine grinding with successive fresh solutions of ferric sulphate, decanting and washing, making alkaline with lime and then treating with successive fresh solutions of 0.3% KCN, it was found possible to extract over 90% of the gold and silver and to obtain about 50% of the copper in the iron sulphate liquor from which it could be recovered by electrolysis with the re-formation of ferric sulphate to be used again in treatment. At the present rates for freight and smelter treatment, however, there was no gain to be obtained from local treatment of concentrate of such high grade and all thought of cyaniding concentrate at El Tigre has been abandoned for the present.

The total metallurgical recovery from the milling ore by sorting at the breaker plant, concentrating, and cyaniding the tailing, is about 93% of the gold and silver, of which from 62 to 64% is recovered in the form of concentrate and sorted high-grade ore.

CHAPTER XIII.

CYANIDE PRACTICE AT THE VETA COLORADO, PARRAL, CHIHUAHUA.

BY BERNARD MACDONALD.

Construction and Plan of Plant.

***Design of Plant.**—The plant of the Veta Colorada M. & S. Co., at Parral, Chihuahua, Mexico, was designed in 1906, and, in accordance with the general scheme of cyanidation plants of that date, provision was made for separate treatment of the pulp as sand and slime. Orders for the machinery were placed with different manufacturers, and the work of grading the site and construction commenced in 1907. The erection of the mill was then placed under the charge of a superintendent, who made certain changes in the design of the machinery and the arrangement of the cyanidation department, having in view its conversion to an all-sliming plant. Toward the end of 1907, on account of the panic then prevailing, the company suspended all work. Some of the machinery had then reached the ground; some was partly erected; some was en route, and a large part was still in the shops in the process of manufacture.

The suspension continued until February, 1910, when I was employed to complete the erection of the plant and put it in operation. My instructions implied authority to purchase such new machinery and make such re-arrangements and modifications of the previous designs in erection as would be necessary to complete the plant and make it suitable for the then recognized improved system of operation, but to retain and use, as far as possible, the machinery previously ordered. After numerous delays, the machinery reached the ground, was erected, and the plant went into operation in February, 1911.

Source and Character of the Ore.—The ore supply in sight consisted of the old dumps at the company's mine, amounting to about 50,000 tons, an indefinite quantity in the mine workings which were caved, and an indefinite quantity of custom ore promised from neighboring mines. The average analysis obtained from the dumps and the old workings is shown opposite:

*From *Mining and Scientific Press*, July 6, 13, 1912.

	Per cent.		Per cent.
SiO_2	69.00	Pb	0.03
Fe_2O_3	3.60	Mn	Trace
Al_2O_3	9.00	Cu	None
Zn	0.01	Ag (ounces)	12.5
Au	Trace		

The silver occurred mainly as sulphide in the ore, but subordinately in the form of chloride and bromide.

Yard and Operating Facilities.—At the head of the mill a large patio or yard was excavated where the custom, dump, and mine ores are received. Tram tracks from the mine, dump, and railroad terminal converge on this yard, and from it a cross-track was laid down along the entire length of the mill with platforms at the various floor-levels of the mill and cyanidation plant for receiving supplies. The company's business and assay offices, warehouse, carpenter shop, and track scales are built on this yard. The coal-bins are situated immediately below the yard-level, and from them the coal is lowered by gravity tram to the power house.

Power.—Electric power generated by steam in a plant on the ground is used for lighting and operating the machinery throughout the mill, with sufficient power to spare for the mining operations. Steam is generated in Heine safety and Fairbanks water-tube boilers, and the electric generators are operated by Harris-Corliss engines, with the generators mounted on the fly-wheel shafts.

Mill Machinery and Equipment.—Below the level of the patio is the receiving bin for all classes of ore, from which it is fed over grizzlies (60-lb. rails inclined 42 degrees and spaced 4 in. apart) to the primary Blake rock-crusher of Australian type with jaw dimensions 12 by 24 in., set to crush to 4-in. size. The product of this crusher joins the ore that passed through the preceding grizzly and gravitates over another leading to the secondary crusher, which has its bars spaced 1½ in. apart and set on an incline of 42 degrees. The secondary crusher is of the same kind and type as the first, with jaw openings 9 by 15 in., set to crush 1½ in. size. The product from this crusher joins that which passed through the preceding grizzly, all falling into a small steel-hopper collecting bin and automatically feeding upon an upwardly inclined belt-conveyor 16 in. wide and 30 ft. long, which, with a travel speed of 150 ft. per minute, delivers the ore into an open-bottom hopper from which it is spouted to a vertical elevator which carries it up to a hopper-bin set above the cross-conveyor built over the battery-bin. The vertical elevator is of the belt and bucket type, 18 in. wide by 40 ft. between centres with buckets 16 by 7 in., set 18 in. apart. The cross con-

veyor-belt over the battery-bin is 18 in. wide by 120 ft. long and is provided with an automatic traveling tripper which discharges the ore into any of the compartments of the battery-bin. This bin is flat bottomed, 12 ft. wide, 20 ft. high, and 120 ft. long, partitioned off for each 10 stamps, and has a total holding capacity of 1,300 tons with a run-off capacity of 1,000 tons. Parallel with the cross conveying-belt a tram track runs over the bin. On this lime required in the treatment is brought and it is mixed with the ore as the latter is delivered into the battery-bin.

Sampling Mill Equipment.—From the stream of ore falling from the collecting hopper at the delivery end of the inclined conveyor-belt two buckets suspended at equal distances apart between two endless chains cut out the sample by taking all the stream for the moment they are passing under the collecting hopper. In operation these buckets are carried over and discharge automatically into a 5-ton bin with 45 degrees bottom, sheeted throughout with steel plate. This is the first cut-out sample. When desired the door of this bin is opened and the contents drawn off and fed to a 7 by 10-in. Dodge crusher set underneath. The crushed product falls on steel plates, where it is shoveled and quartered, the reject sent to the vertical elevator and by it delivered to the battery-bin, while the retained sample goes to a pair of 12 by 12-in. fine rolls, the product from which falls on the floor and is shovel-sampled, the reject being handled as before. The retained sample is then sent through two laboratory grinders in sequence, being shovel-quartered after each grinder and the final sample put through an Iler pulverizer, the entire product of which is quartered on oil-cloth and sent to the assay office. The cutting-down floor underneath the sampling machinery has an area of 12 by 20 ft. and is sheeted with ¼-in. steel plate throughout.

Stamps and Dewaterers.—From the battery-bin the ore is fed to the stamp-mortars by suspended Challenge feeders; the mortars weigh 10,500 lb., and are bolted to concrete foundations with a ¼-in. rubber sheet intervening. The stamps, of which there are 80, weigh 1,050 lb. each and are set to drop 7½ in. 104 times per minute. The depth of the discharge is 4 in. above the dies, about 1 in. above the ore-bed. The ore is crushed in the precipitated solution returned from the zinc-boxes and fed to the mortars with the ore in the ratio of 8 tons to 1 ton of the dry ore. Rectangular screens of the equivalent of 14 to 16 mesh were used, and 35% of the pulp issuing from the motars would pass 200 mesh. The crushing capacity is about 5 tons per stamp.

On issuing from the mortars, the pulp from each 20 stamps is piped to a 4 by 4-ft. dewatering cone which overflows 20% of the battery water with a varying quantity of 200-mesh pulp in suspension, this overflow being piped direct to the Dorr thickeners.

The underflow pulp from these dewatering cones, containing between 6 and 7 tons of solution to 1 of solids, is carried in floor (cement) launders to a collecting tank from which it spouts to two belt-bucket elevators which lift and deliver it to a distributing box set 20 ft. above the floor. From this box it is equally distributed and carried in three distributing pipes to three duplex Dorr classifiers each 5 by 15 ft. The slime portion of the pulp classified by these, of which about 80% goes through 200 mesh, goes direct to the Dorr thickening tanks, while the sand classification is distributed in pipes to the scoop-boxes of the tube-mill.

Fine Grinders.—There are five tube-mills set together, each 5 by 14 ft., operated at from 27 to 30 revolutions per minute. The discharge from all these mills is received in a cross-launder set on 10% grade, which in turn discharges into the boots of three belt-bucket elevators, two of which are kept in operation and one held in reserve. These elevators raise the tube-mill product to a distributing box from which it is led back through pipes to the Dorr classifiers, where it is received for re-classification along with the battery pulp underflowing from the dewatering cones mentioned above. Thus, the classification of the pulp, with the exception of that overflowing from the dewatering cones, is effected in a closed circuit and results in a slime classification going to the thickening tanks, 80% of which is of −200 mesh.

Pulp Thickeners.—The three Dorr thickening tanks which receive the classified slime are 36 ft. diameter by 12 ft. high, and in them the collecting rabbles are geared to make one revolution every seven minutes. The clear solution overflowing from these tanks goes to a 12 by 36-ft. sump-tank, from which it is pumped back to the two head-supply tanks (set 30 ft. above the batteries), each of which is 12 ft. high by 36 ft. diameter. When this separated solution from the thickening tanks becomes sufficiently charged with silver to warrant precipitation, provision is made to pipe it to a filter-press through which it flows by gravity and is clarified and then goes to the zinc-boxes for precipitation.

Dilution and Elevation of the Thickened Pulp to the Treatment Tanks.—The thickened pulp underflow from the thickeners, containing about 1½ of solution to 1 of solids, is piped to the boots of two elevators by which it is raised to the agitation tanks. As this pulp is delivered to the elevators it is diluted by precipitated solution returned from the zinc-boxes mixed with the wash-water from the filter-presses to the consistence of 2:1, which was found to give the best results in treatment.

The two elevators lifting to the agitation tanks are of the belt-bucket type, 55 ft. between centres, the belts being 24 in. wide. One of these elevators is in constant use and the other held in reserve. The elevated pulp is received in a box, at the

Fig. 28. Veta, Colorada, General View.

head of the elevators, fitted with a cover and false bottom of wire screen, into which the cyanide and lead acetate required to bring the solution in the pulp up to treatment strength are placed every hour, and are gradually dissolved by the splash of the discharging pulp. Provision is made for piping the pulp from the receiving box to any of the agitation tanks for treatment by the individual-tank process, or for delivery into the first of the series of agitation tanks for treatment by the continuous process as desired. In the latter case, which was permanently adopted after extended trial, provision is made for the pulp to flow from tank to tank and be drawn from the last of the series to the filter-presses.

Agitation Tanks.—The treatment or agitation tanks consist of a battery of six tanks which contains one standard Pachuca tank, 15 ft. diameter by 45 ft. high, and five Parral tanks 25 ft. diameter by 42 ft. high, having holding capacities of 83 and 250 metric tons, respectively, of 2:1 pulp. The agitation is effected by compressed air in both tank systems; in the Pachuca tank in the usual way through a central lift-pipe of 16-in. diameter, and in each of the Parral tanks by four lift-pipes each of 12-in. diameter, set equidistant from each other and 2½ ft. from the interior side of the tank. The discharge ends of the lifting pipes in the Parral tanks are set horizontally and so directed that the discharging pulp flows in the same direction as segmental cords with respect to the interior side of the tank. The force of the discharge pulp sets up and maintains a rotary flow in the entire pulp charge from top to bottom of the tank. This rotary flow preserves the solution and solid constituents of the pulp charge in proper proportional mixture and prevents the settlement of the pulp on the bottom of the tank into dead accumulations.

Filter-Presses.—When the treatment cycle in the agitation tanks is completed, the pulp is drawn off to a battery of Kelly filter-presses, of which there were 8 operated hydraulically from a central platform in two units of 4 each. Each of the presses is 5 by 15 ft. and contains 13 leaves having a total filtering area of 1,500 square feet. The Kelly filter-presses, and the excess-pulp, precipitated-solution, and wash-water tanks are shown in the foreground in the illustrations.

The operating cycle and results are as follows: Density of pulp received, 1.26; percentage of - 200 pulp, 80; time charging and building cake, 13 to 28 minutes; expelling surplus pulp, 2; washing with precipitated solution, 10; washing with water, 3; drying with air, 3; discharging cake, 10; total cycle, 40 to 50 minutes; thickness of cake ⅝ to 1 in.; moisture in discharged cake, 15 to 18%; assay difference between washed and unwashed cake, 3 gm. silver.

Disposal of the Cake.—The cake discharged from the filter-presses falls into a V-shaped box sheeted with steel plates which

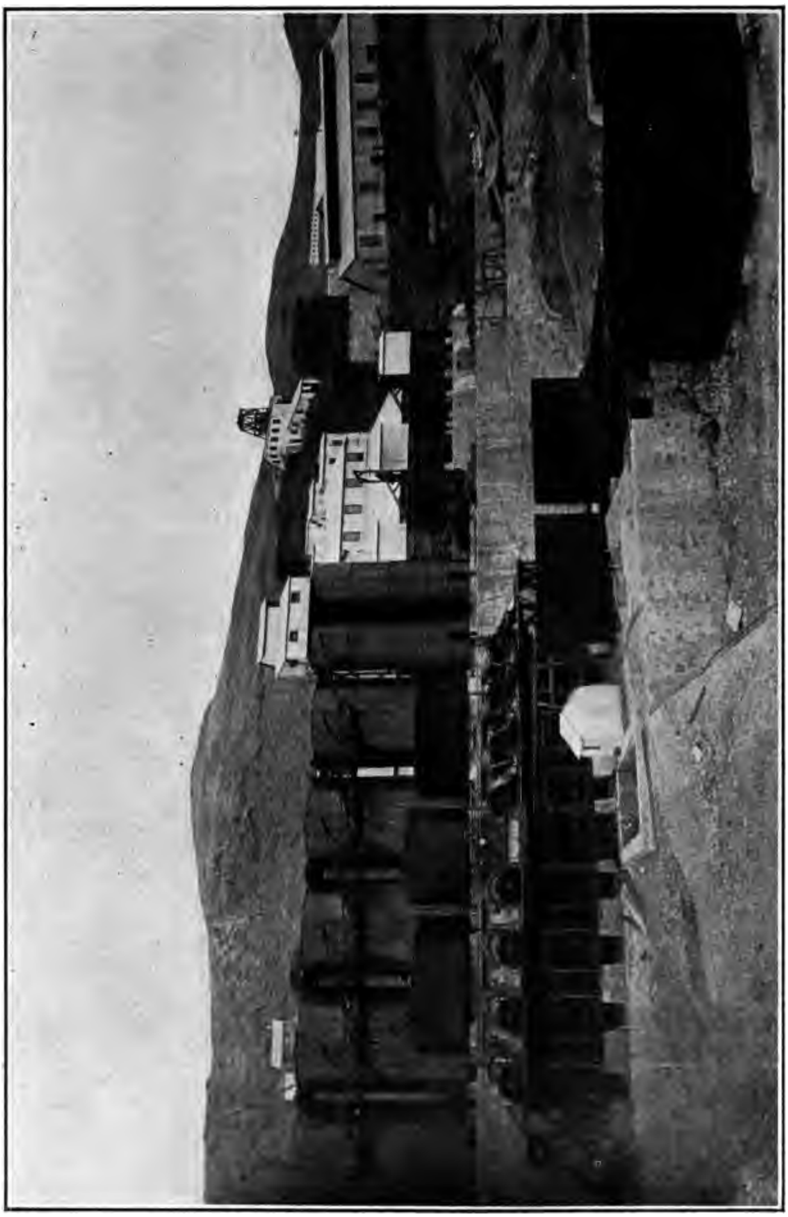


Fig. 29. Agitators and Filters.

extended transversely underneath all the presses. In the bottom of this box is a right and left-hand screw-conveyor which works the discharge cake, mixed with water, to a central box where it is cut up with a chopper made of spikes inserted in a hub to the shaft of the screw-conveyor. As the cake is being chopped it is struck by a 1-in. stream of water under pressure-head of 200 ft. and thus mixed and diluted, it passes through the tailing launder, set on grade of 12%, to the tailing dump.

Manipulation of the Filtered Solution.—The filtered rich solution from the presses is collected in a launder underneath, from which it flows in a pipe to two collecting tanks each 10 ft. high by 20 ft. diameter, from which it flows to five clarifying boxes, each having five compartments. These boxes are of the zinc-box type with baffle partitions between the compartments and 3 by 15 ft. over all by 5 ft. deep. Leaves of cocoa matting are placed in the compartments and spaced 2 in. apart. The outflow from these boxes is piped to two storage tanks, each 36 ft. diameter by 10 ft. high, from which the flow to the zinc-boxes was so regulated as to be continuous and uniform.

Measuring the Solution.—These tanks were designed to be used alternately in feeding the zinc-boxes; when one is feeding the zinc-boxes the other is receiving the solution flowing from the clarifying boxes. With floats connected by ropes run over sheaves to weights which rise and fall along vertical recording boards in the zinc-room like those of railroad tanks, the zinc-room man is enabled to record the cubic feet or tons of solution that passes each shift of 12 hours through the zinc-boxes.

Sampling the Solution.—The solution coming to the zinc-boxes is sampled by a drip-cork tapped into the main delivery pipe, and thus the tonnage of solution going through the zinc-boxes, and its assay value, is recorded. In like manner the main pipe carrying the precipitated solution from the zinc-boxes to the precipitated solution sump-tank is provided with a drip sampler, the assay of which subtracted from the head sample shows the amount precipitated in the boxes.

Return of the Solution.—The sump-tank is 10 by 36 ft., and the precipitated solution received in it is pumped back to an intermediate sump-tank from which the amount required for diluting the pulp going to the elevator lifting to the agitation tanks is supplied, the amount not needed for this purpose being pumped back to the supply-tanks at the head of the mill.

The Precipitation Room.—The precipitation of the rich solution is effected by zinc shavings in 11 zinc-boxes, each box having five compartments, 3 by 3 by 3 ft., for the zinc shavings and having a total holding capacity of 1331 cu. ft. of shavings. As a rule only four compartments of each box were charged with shavings,

since the precipitation is complete in four of the compartments. From the bottom of these compartments, which are pyramidal in shape, a pipe with stop-clock is provided for the discharge of precipitate during the clean-ups, which are made weekly. The precipitate, with the associated zinc shorts, as discharged from the zinc-boxes, is flushed and brushed through half-round steel launders to a collecting sump-tank provided with screen trays at its top, through which the precipitate passes. The zinc shorts, + 60 mesh, in the precipitate are caught in the screening trays, where they are scrubbed and washed and removed from time to time, and returned to the head compartments of the zinc-boxes, where they are gradually consumed by the flow of the rich solution entering for precipitation.

Filter Pressing the Precipitate.—From the sump-tank the precipitate is pumped through a Dehne press, 8 by 2½ ft., with 30 leaves having a total filtering area of 650 sq. ft. The effluent solution from the precipitate press goes to the zinc-boxes, where any escaping precipitate is caught in the zinc shavings.

Fluxing and Drying the Precipitate.—When the cake of precipitate is built in the press it is discharged into a shallow box run on flat wheels underneath the press. In this box the precipitate when sampled for moisture is weighed, mixed with the required fluxes, and filled into iron trays 30 by 10 by 4 in. and placed on the shelves of a steam-drying cabinet. This is a closed cupboard-shaped box made of 3-16 in. steel plate, measuring 8 by 6 by 3 ft., with shelves made of 1¼-in. pipe extending from 3-in. headers at the sides, through which low-pressure steam generated in an upright 5-hp. boiler circulates. The cabinet can be securely locked and serves as a safe for the precipitate until needed for melting.

Melting and Sampling.—For melting the precipitate there are three Steele-Harvey tilting oil-fired furnaces, containing No. 275 Monarch crucibles, into which the dried and fluxed precipitate is charged and the melt effected. The bars weigh from 70 to 75 lb. avoirdupois each, and are sampled by boring at opposite corners of both sides 3-16 in. holes to the depth of 1 in. The silver in the bars ranges from 850 to 900 fine.

Air-Compressors.—An Ingersoll-Rand, type 10, air-compressor generates the compressed air used for agitation, filter-pressing and air lifts. This compressor has a sea-level displacement of 1000 cu. ft. of free air per minute, and at the plant, 6000 feet elevation, about 80% efficiency is obtained. The pressure carried for this plant is 30 lb. per square inch.

Motors.—Westinghouse motors are used throughout the plant, the type being 3-phase, 25 cycle, 440-volt, alternating current. Their number, where used, rated horse-power, hours run per day, and power consumed is as opposite:

No.	Used to run	Rated hp.	Consumed hp.	Running hr. per day
1	Two Blake crushers and belt bucket-elevator	50.0	49.7	10
1	Cross conveyor-belt	5.0	2.7	10
1	Sampling mill machinery	15.0	5.0	8
4	Stamp batteries, one to each 20 stamps	200.0	198.8	24
3	Three Dorr classifiers	5.0	3.0	24
1	Five pulp elevators	20.0	12.7	24
5	Five tube-mills, 40 hp.	200.0	155.0	24
3	Three Dorr thickeners	3.0	1.0	24
1	One belt bucket-elevator to treatment tanks	10.0	4.2	24
1	Ribbon pulp conveyor	10.0	5.3	20
1	Triplex pump returning solution	20.0	16.8	20
1	Triplex pump returning solution	15.0	7.6	20
1	Triplex pump returning solution	7.5	2.0	20
1	Pump and lathe in zinc room	5.0	5.7	10
1	Ingersoll-Rand compressor	100.0	83.3	24
1	Motor generator	40.0	32.0	24
	Lighting (417 hp.-hr.), from plant	17.4	17.4	12
28	Totals	722.9	602.4	

From the table it will be seen that the 28 motors connected with and driving the milling machinery, have a total rated capacity of 722.9 hp., and that these motors when carrying the full load of all machinery running together, including the pro-rated lighting, consumed 602.4 hp., 83% of the nominal power of the motors, or say, 1.5 hp. per ton of milling capacity.

Comments and Criticism.

I have so far given an outline of the flow-sheet or travel of the ore pulp and solution in treatment through the mill of the Veta Colorada M. & S. Co., at Parral, Mexico. I propose now to submit comments and criticism of the millsite, machinery and practice.

The Millsite.—The selection of a site for the erection of a milling plant is of fundamental importance, and the greatest care, considering all the factors bearing favorably or adversely on the economy of future operations, should be intelligently and carefully exercised before making the final location. The same is equally true and of only relatively less importance with respect to the sites of all appurtenant buildings such as power-house, machine-shop, precipitation-house, warehouse, and assay office. Before the grade stakes are finally set, the designing engineer should have a clear comprehension of all the factors commercially affecting future operations, and from them work out a favorable balance for the site to be chosen. The selection of a site that is ideally perfect is impracticable, even in the most favorable localities, but existing conditions should be so weighed and balanced that the best site

possible shall be chosen. The preliminary investigation and study necessary for this purpose is not always given by the designing engineer. It frequently happens that the site is arbitrarily selected by those who have little or no experience in mill construction or operation, and the engineer is left to do the best he can with the site so selected. Be this as it may, the fact remains that serious errors in the site of the mill or the accessory buildings exact their toll from the beginning of operations.

These reflections have been inspired from observations covering milling plants in general and not alone from the site selected for the Veta Colorada mill, although the site of the plant was not the best that could have been selected. The site for this mill was chosen too far down the hillside (a mistake frequently made), whereas, with a few minor disadvantages, completely overshadowed by general improvement in facilities for construction and operation, the mill might have been placed from 50 to 100 ft. higher. Such a site would have saved considerable expense in grading and retaining walls for the mill benches and in the disposal of the tailing of the ore treated, and the plant would have been more accessible for construction and operation.

In the design of the plant the old idea of placing the precipitation-room at the lower end of the mill was adopted. The result of this, coupled with the necessity of securing flow grades on the hillsides for the several benches of the plant, caused it to be stretched out so that it occupied a length of over 500 ft. from the crusher-bin to the precipitation-house. Notwithstanding this length, the site for the precipitation-house, and the sump tank and pump-house adjoining measuring 80 by 150 ft., had to be blasted out of the solid rock to an average depth of 12 ft., and, to provide floor drainage, a small tunnel had to be run from the bottom of this grade to a nearby creek. The precipitation-house, as designed and built, was 75 by 100 ft., just twice the size required, and at the place selected it was the most distant of all of the mill buildings from the offices and living quarters of the American employees, which made its supervision and protection more difficult.

In completing the plant, I seriously considered abandoning the use of this building for precipitation and the erecting of a new one on the patio at the head of the mill, where it would have had all the advantages of being close to the company's offices and the living quarters of the American employees, but as the house had already been substantially built of masonry, floored with cement, and the sump tank and return pump erected, the expense deterred me from doing so. Had the mill been placed higher on the hillside, the steeper contour would have provided for closer building of the several plant units, saved hundreds of feet of piping, and thousands of dollars in grading, besides affording better accessibility for supervision and more economical conditions for operation.

Power Plant.—The placing of this plant, and the design of its machinery could have been improved. It was built close to the mill, to which fuel had to be hauled from the railroad terminal, and the boilers supplied with mine-water carrying deleterious solids in suspension, while, with a 7-mile transmission line, it could have been built at the junction of two railroads where fuel would have been cheaper and plenty of clear water for the boilers available. And, for some reason hard to explain, the electric generators and the mill motors were designed for a 25-cycle current, a design at variance with common commercial practice. A 60-cycle electric plant was in existence at the time, designed for the sale of current to users in the district, and, at the present time, a 15,000-hp., 60-cycle, hydro-electric plant is bringing in its lines to the district to dispose of its power to the mines and mills. This power cannot be used in this plant without the intervention of "frequency changer" and a corresponding consumption of power.

The Rock-Breakers.—As already stated, these are of the 'Austrian type,' which in operation close their jaws and crush the rock on the down-throw of the pitman, instead of on the up-motion as in the ordinary type. This antipodean type may work satisfactorily in Australia, but as designed for this plant it did not do so. The force exerted in crushing the rock on the down-thrust of the pitman reacted upwardly on the bearing boxes of the fly-wheels, loosening and frequently breaking the stud-bolts that held down the caps. The keeping of the toggles in adjustment was also much more difficult than with the ordinary type of rock-breaker.

Mortar Foundations.—These were built of concrete in separate blocks for 20 stamps at some time before work was suspended in 1907. I found these blocks cracked horizontally from end to end, the cracks being 6 to 12 in. apart, so that, at places, the top layers could be easily lifted by driving a gad in the cracks. I puzzled over this problem for some months pending the arrival of the mortars from the factory, and finally reached the conclusion that these cracks were due to one or more of the following causes: (1) the concrete was laid in layers and each layer allowed to dry before the next one above was laid down, the junction of the two contiguous layers constituting a line of weakness along which the unequal shrinkage of the dry and wet concrete was manifested in the cracks; (2) the sand in the different layers of concrete differed in character and quantity; (3) the frequent but erroneous practice of putting a thin leveling layer of neat cement on top of the block, instead of laying the block a trifle high and bringing it to proper level with a chipping tool. It was evident that the foundation blocks would crumble under blows at the stamps unless something was done to make the layers cohesive, for even the vibration of the blow of a sledge at one end of a block could be plainly felt at the other. To dig out the foundations and put in new ones, it was estimated,

would have cost $5000. This would have been safe, but that portion of the mill building was roofed and sided with corrugated iron and would have had to be stripped, as blasting would have been necessary in order to get out the deeper sections of the blocks. I finally concluded to have the blocks drilled with vertical holes to the depth of 30 in., 30 holes under each two mortars and the shoes of the battery posts, and dropped into each hole a piece of 7/8-in. octagon drill steel 30 in. long, around which was poured melted sulphur until the hole was filled level with the top of the block. After this there was no vibration felt under the blows of the sledge-hammer, and the rubber pads were put on and the mortars bolted down. On two of the blocks the top leveling layer of concrete ranging from 1/4 to 1½ in. crumbled off. From these I removed the rubber pads and wedged up the mortars to the level and tamped the space between them and the concrete solid with 'rust joint.' The result was satisfactory, as none of the foundation blocks showed any signs of weakness under operation.

Concentration.—The mill is equipped with 24 concentrating tables, which were put into service when operations began. Owing to the fact that the dump ore to be treated was the ferruginous oxidized low-grade material discarded from the ore previously mined from the upper levels, and that most of the other ore treated in the mill was of the same character, it was not practicable to make a clean grade of concentrate that could be profitably shipped to the smelters. After tests showed that better commercial results could be obtained from cyaniding the pulp without concentration, the tables were left idle and the pulp treated entirely by cyanidation. The concentrators remain in place ready to be started if complex sulphide ore is found in the deeper levels.

Tube-mills.—These measure 5 by 14 ft. and were originally designed to be driven by 30-hp. motors, but before starting I substituted 40-hp. motors, and even these had barely enough power to start the loaded mill from rest without reverse swinging. The spur and pinion gearing by which the motors operated the mill, proved not to have sufficient strength for the strains of operation, and after a couple of months' use three of these broke and were replaced by belt pulleys. The motors, when connected by belts, started the tube-mills with greater ease than when previously connected by spur-gearing.

It will be interesting to millmen to know of a little device by which I improved the grinding capacity of the tube-mills and lightened their running load. The discharge openings of the tube-mills were 7-in. diameter and the pebble charge came up to the bottom of these openings, or 3½ in. below the longitudinal centre of the mill when at rest. I reasoned that if the discharge opening was smaller, so the mill could be loaded with pebbles until the longitudinal centre line of the mill was reached, this load would be easier

started and better grinding would be done. To test this idea, I had a wooden plug 8 in. long turned on the lathe to the exact fit of the discharge opening and a 1½-in. hole bored through the centre line of the 8-in. dimension. This plug was driven snugly into the discharge opening and the mill filled with pebbles an inch or so above the centre. The result was very gratifying, the mill was easier to turn over from rest and to run than with the lower charge of pebbles, and the grinding was better. Since this time I have recommended this in other mills, where it showed similar favorable results.

Agitation Tanks.—As stated at the beginning of these notes, the plant, as originally designed, provided for the dual treatment of the pulp as sand and slime. For the agitation of the slime, tanks 25 ft. in diameter by 12 ft. high equipped with stirring arms were provided in the original design and in 1907 two standard Pachuca tanks 25 ft. in diameter by 45 ft. high were purchased. Most of the material for these tanks had reached the site, but none of them was erected.

For some years previous I had been developing and experimenting with a system of pneumatic agitation with the object of overcoming the defects of the Pachuca tank system, namely: (1) The uneconomical shape of the tank, having small diameter and great height, which provided small holding capacity per pound of steel employed in the construction; (2) the high consumption of compressed air for the work done, and the trouble and expense for operating the rubber sleeve air nozzle; (3) the vertical settlement of the pulp immediately around the central lift-pipe and the consequent imperfect mixture of the solid and solution constituents of the pulp charge around the side of the tank.

By experiments based on scientific principles, I had developed a system which worked out satisfactorily in a small way and promised to eliminate the defects above mentioned, and now the opportunity of applying it on a large commercial scale was presented. But what were to be the dimensions of the first 'Parral tanks' to make the best compromise of the various conditions existing at this plant? The factors to be equated in the compromise were: (1) my former ideal of an agitation tank to be of such dimensions as would give the greatest holding capacity per pound of steel used in its construction; (2) the height of the two standard Pachuca tanks, to be units in the series, the steel plate for which was in stock; (3) a large quantity of steel plate shaped for the original agitation tanks 25 by 12 ft. which was in stock; (4) arrangement of tanks suitable for either the individual or continuous processes of treatment as would prove best under experiment; (5) the original design to make the plant's capacity 500 tons per day to be carried out so far as the machinery and material in stock would permit.

The compromised design and dimensions of the agitation tanks under these conditions resulted in a battery of tanks consisting of the

two Pachucas, each 15 ft. in diameter by 45 ft. high, and five Parral tanks, each 25 ft. in diameter by 42 ft. high. The Pachucas had a holding capacity of 83 metric tons each and the Parrals 250 metric tons each; the pulp consisting of two parts by weight of solution to one of the dry solids. With the exception of one of the Pachuca tanks, which was set apart as a storage tank for the wash-water required for the filter-presses, these tanks were 'piped' top and bottom so as to be operated by either the individual or continuous agitation systems as might be demonstrated by test to be preferable. The Pachuca tank was equipped in the usual way with the central transfer pipe, 16 ft. diameter, provided with the 1½-in pipe with rubber sleeve nozzle for the introduction of compressed air as furnished by the manufacturers, and the Parral tanks were equipped with four 12-in. transfer pipes, each of which was provided with a 1-in. compressed-air pipe equipped with a ball-valve nozzle. Operation was commenced and continued one month by the continuous process of agitation treatment and then changed to the individual process. So far as the extraction went, there was no difference shown by either process over the other, but on account of its greater economy and simplicity of operation, the continuous process was adopted after the tests.

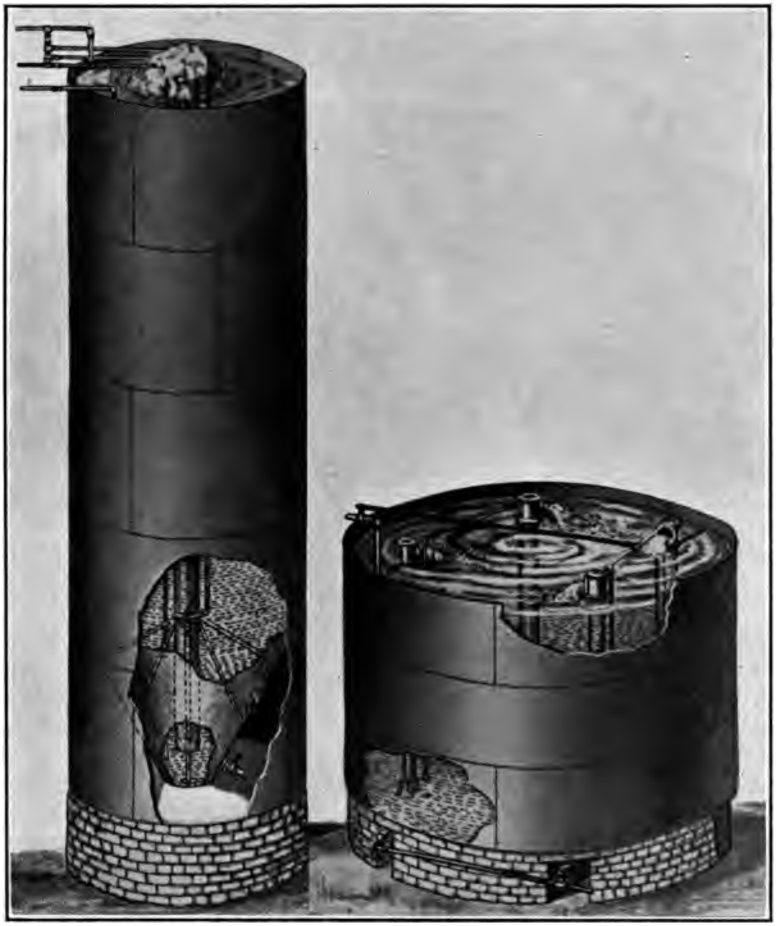
The Pachuca and Parral Systems.—The Pachuca tank here gave as favorable results as usual, and, in comparison with the Parral tanks,

> "I could have been happy with either,
> Were the other dear charmer away."

However, in this case, there was a decided difference between the charmers. So far as it was possible to determine the amount of air consumed in the tanks operating side by side, treating pulp of the same specific gravity, by the degree of valve openings on the compressed-air supply pipes of the same diameters, there was no greater consumption of air in agitating the Parral tank holding 250 tons than in agitating the Pachuca holding only 83 tons. Besides, the rubber sleeve nozzle in the Pachuca tank gave considerable trouble in starting up agitation after the compressed air was shut off for any reason, while, in the Parral tanks under the same condition, the ball-valve nozzles would start up agitation promptly without trouble. After the month's run, when the change was made from the continuous to the individual system of agitation, the rubber sleeve was found to be frazzled and to have lost its elasticity. At this time the 1-in. air-nozzle ball-valve was attached to the end of the 1½-in. pipe in the Pachuca tank, and afterward there was no more trouble in starting and the agitation was equally active, with appreciably less valve opening. The following letter from D. S. Colland, assistant to C. W. Van Law, director for the Compania de Real del Monte y Pachuca, will be interesting as

corroborative evidence on this point. It relates to experience at Pachuca in February, 1912:

"Just a word regarding the installation of ball-valves in our Pachuca tanks. Before the receipt of your letter, accompanied by the 1-in. valve, we had already tried out the ¾-in. that you had



Pachuca Tank. *Fig. 30.* *Parral Tank.*

left here. As you know, the valves we had always used are the rubber sleeves outside of 1½-in. air pipe. This pipe was reduced at the bottom of the tank for the ¾-in. valve and all the air turned on at the 1½-in. valve at the top. This resulted in the rising of the pulp in centre column but 2 in., the air bubbles

simply forcing their way through the thick slime. Upon receipt of the 1-in. valve it was connected in the same manner and it worked excellently. The 1½-in. air valve at top of tank is opened about three-quarters of one turn for all the rubber sleeve valves, and the ball-valve has been operating now nearly a week, giving entire satisfaction. On last Friday the power was interrupted several minutes. The four tanks operated with rubber valves had to be opened full to start them, while the valve on the tank equipped with ball was not touched. As soon as the air pressure came on, this tank began agitating exactly as before interruption. The slime during the test has been approximately 1.7 to 1."

The Parral system of agitation, like the Pachuca, is effected by the continuous transfer of the pulp from the bottom to the top of the tank by air-lift, but the details of the method of operation by the two systems are entirely different. In the Pachuca, which is fitted with a cone-bottom, the agitation of the pulp is effected by its continuous transfer from the bottom to the top of the tank, through a pipe 16-in. in diameter fixed centrally in the tank with its intake near the apex of the cone at the bottom and the delivery end at or near the top of the pulp charge. The compressed air is delivered to the intake end of the transfer pipe through a 1½-in. pipe capped at the end and perforated on the sides for a length of 6 to 8 in. back from the cap. To prevent the pulp from entering the perforations, a tight-fitting rubber sleeve is drawn over the end and clamped above the perforated section. When the compressed air is turned on, the rubber sleeve expands against the hydrostatic pressure surrounding it, and the air passes out under it and, entering and ascending through the transfer pipes, carries the pulp from the bottom and discharges it at the top of the tank. The mechanical defects which I found in the Pachuca tank and system of agitation will come under the following heads:

1. The tall narrow shape of the tank with its comparatively small holding capacity per pound of steel and its high cost of construction.

2. The cost of elevating the charge to tall tanks during operation, barring plants erected on exceptionally steep millsites.

3. The imperfections of the means employed for admitting the compressed air to the lift-pipe, and excluding the pulp from choking the air-pipe during times when the air is shut off.

4. The excessive amount of power necessary to generate the high pressure and volume of air required to effect the agitation in a central lift-pipe of large diameter.

5. The imperfect mixture of the solid and liquid constituents of the pulp in the tank during agitation.

6. The difficulty of starting agitation when compressed air has been shut off and the solids in the pulp settled in the cone at the bottom of the tank.

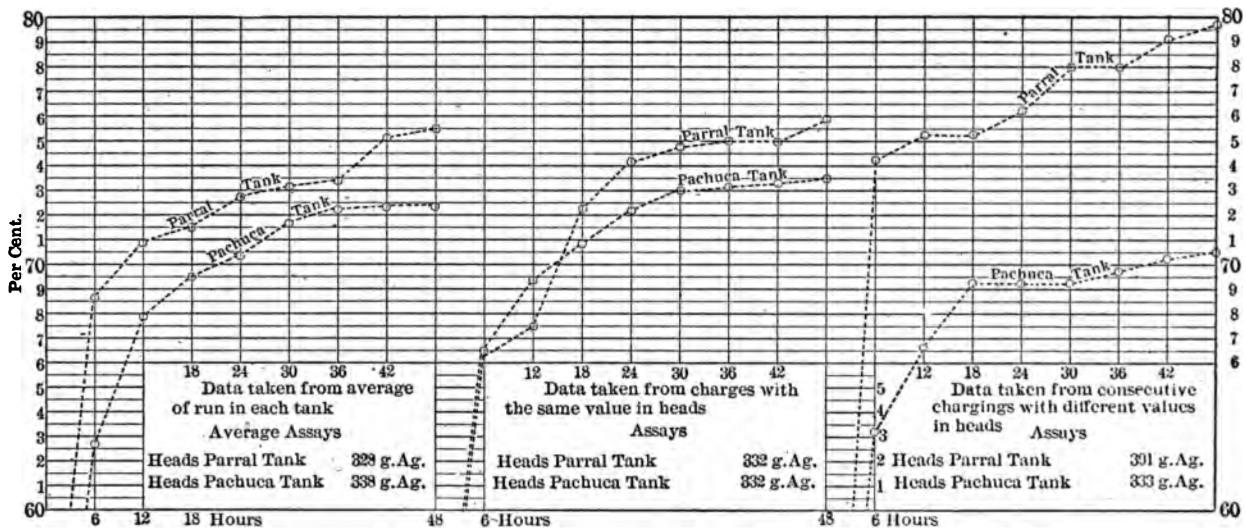


Fig. 31. Extraction-Curves Plotted from Records of Operations in Pachuca and Parral Tanks. Samples taken every six hours and assayed.

In the Parral tank system these defects have been eliminated by the use of a plurality of transfer pipes of comparatively small diameter, the number of these pipes being in proportion to the diameter of the tank, which may be economically proportioned to its height so as to give the greatest holding capacity per pound of steel or material used in the construction. The accompanying illustration and table of comparison will indicate some of the advantageous features in construction of the Parral type of tank.

Comparison Table of Corresponding Items in Standard Parral and Pachuca Tanks.

Points of Comparison.	Parral.	Pachuca.
Height in feet	15	45
Diameter in feet	25	15
Horizontal area in square feet	490.8	176.7
Effective holding height in feet	14	39
Holding capacity in cubic feet	7671.2	6891.3
Holding capacity in metric tons of solids:		
Pulp ratio: solution 2, solids 1	92.8	83.3
" 1½ " 1	139.4	125.3
" 1 " 1	155.3	139.5
Weight of steel plate and all construction:		
Material in pounds	14,650	33,000
Lb. steel per ton of 2.1 pulp	157	400
Lb. air-pressure required for agitation	8 to 10	30 to 50

I have selected the dimensions 25 by 15 ft. as the standard size for the Parral tank for the following reasons:

1. A tank of these dimensions makes a convenient unit for most plants, as it holds from 90 to 100 metric tons of dry pulp of a specific gravity usually suitable in most ores for treatment.

2. The high holding capacity per pound of material in its construction.

3. Its comparatively low height, which makes charging possible by gravity flow in most hillside plants, and permits agitation by compressed air of 8 to 10-lb. pressure per square inch.

However, the system by which agitation is effected may be installed in and will work well in tanks of any diameter and any height. In the plant of the Veta Colorada M. & S. Co., the tanks were 25 ft. in diameter and the height was built up to 42 ft. to correspond with the Pachuca tanks included in the battery, as the walks on top and sampling platform could be more conveniently arranged in connection with a uniform height. All the trouble and expense due to the short life of the rubber sleeve as a nozzle on the compressed-air pipe and the high consumption of air required to operate it (the total hydrostatic pressure of the tank charge on this sleeve being 850 lb.) in the Pachuca tank is obviated by the air-nozzle ball-valve of the Parral tank. This valve is balanced

in the hydrostatic head of the pulp charge except about 50 lb., due to the seat area. These valves have been in continuous operation for 16 months without renewals or showing any sign of loss of efficiency, so their life limit cannot be fixed. With these valves,

Fig. 32. Commencement of Pulp Transfer in Parral Tank.

agitation has been started and brought to normal running in 15 minutes after the air had been shut off and agitation suspended for 24 hours, by simply turning on the air.

The imperfect mixture of the pulp charge of the Pachuca tank due to the vertical settlement of the pulp immediately around the

central transfer pipe, is prevented in the Parral system by creating a rotary flow in the tank charge whereby the whole mass revolves round and round from top to bottom within the tank, and the solids are thus carried spirally in suspension, and therefore the distance of their travel from top to bottom of tank is many times greater than if allowed to settle vertically. The rotary flow is created by the force of the horizontal discharge from the several transfer pipes in the same direction with respect to the side of the tank on the surface of the charge.

The surprising facts in connection with the rotary flow in the tanks are that it is generated by so small a force as the spouting from the discharge of the transfer pipes and that when generated on the surface of the charge it will extend to the bottom of the tank with sufficient activity to prevent the settlement of the pulp in dead accumulations on the bottom such as takes place in tanks 42 ft. high. An analogy, showing the mobility of water under slight force and the persistence of flow when once created in it, is seen in the great ocean currents which, generated by the trade winds, revolve continuously between the shores of continents thousands of miles apart. The Gulf stream is generated by the easterly trade winds blowing on the surface of the Caribbean sea which drive these waters westward before them till they make their escape back again into the Atlantic ocean between Florida and Cuba, where, as is generally known, the flow of the current is 6 ft. per second and its strength such as to sweep the sea-bottom, here 600 ft. below the surface, as clean as a floor and force the current in a rotary path across the Atlantic ocean to Europe and back again. Whether this analogy is correct or not, the fact remains that the rotary flow created at the surface of the charge in a Parral tank maintains a continuous revolution in the entire mass within the tank which keeps the solid and solution constituents of the pulp in proper proportional mixture, in every part of the tank at all times during treatment, while the spouting of the pulp at the surface of the charge gives the aeration required for the chemical reactions.

Extraction.—That the extraction is better in the Parral tank as compared with the Pachuca is shown by the diagram on page 42, which is self-explanatory. The data from which the curves are traced were taken from the log records of the results obtained when both tank systems were being operated by the individual process.

Chemical Constituents of Solution:

KCN	0.15%
Protective alkalinity	850 gm.
Lead	3 gm.

Physical Condition of Pulp:

Dilution	2 to 1
Specific gravity	1.26

Screen sizes: + 200, 15 to 20%, –200, 75 to 80%

Extraction:

Prior to tank treatment, 10 to 20%.
By tank treatment, as per chart.
Total extraction, as per chart × 10 to 20 %.

In explanation of the percentage column of the above chart, it may be said that more than 62% of the metal was extracted during the first six hours' agitation.

The low extraction shown in the curves was due to the character of the dump ores being treated. The dump as it was built up was mixed with accumulation of rotten timbers from the mine workings and the ashes and charcoal from steam plant near the shaft. It contained, besides, the mud and oxidized matter from the ore-washer which was on the dump and used for washing the mine ore preparatory to hand sorting.

The Kelly Filters.—These did satisfactory work on the slime, which was very clayey. The filter test made on the slime of an average sample of the ore before the filters were ordered, showed that under 40 lb. air pressure it was only possible to build a cake 3/8 in. thick in 20 minutes. In regular operation, after the presses were erected, there was no difficulty experienced in building a cake 1 in. thick in the same time and drying it to 15% moisture under 30 lb. air-pressure for 3 minutes. Had water not been available for flushing the discharged cake to the dump, it would have been dry enough at 15% moisture for disposal by belt-conveyor. Having filter capacity to spare, one of the filter-presses was very successfully used as a clarifier for the decanted solution from the Dorr thickening tanks on its way to the zinc-boxes for precipitation. This solution flowed through the leaves of the press by gravity under a head of 25 ft. and came out perfectly clear. About a 1/4-in. cake built up of the fine matter suspended in the solution would deposit on the leaves each 24 hours. This had to be discharged, as nearly all flow stopped when a 3/8-in. cake would form on the leaves.

CHAPTER XIV.

CYANIDE PRACTICE IN SMALL PLANTS.

There are many mines where the installation of a small plant for the cyanidation of the ore would be advantageous, and in order to demonstrate the simplicity of such a plant, the following photograph and description of the installation of the mill of the El Nuevo Rosario y Anexas Mining Co. at Santa Maria Peñoles, Oaxaca, Mexico, has been courteously furnished me by the manager for the company, L. E. Lepine:

The ore treated in this mill contains gold only. It is crushed to 40-mesh by ten 850-lb. stamps, at the rate of 20 tons per day, passes over a Wilfley concentrator, and thence to two cone classifiers, which divide the pulp into slime and sand for proper treatment.

Cyanidation plant.—This plant is shown on left in Fig. 33, and includes:

2 tanks 15 ft. in diameter by 6 ft. 6 in. in height for storing strong and weak solutions.
4 tanks 16 ft. in diameter by 6 ft. high for slime-treatment, using compressed air for agitation.
4 tanks 16 ft. in diameter by 3 ft. high for sand-treatment. Leaching through filter bottom of cocoa-matting.
1 tank 13 ft. 6 in. in diameter by 3 ft. in height, with filter bottom for filtering the solutions decanted from the slime-treatment tanks before passing them to the precipitation-boxes.
1 tank 4 ft. by 4 ft. by 2 ft. 6 in. with filter bottom, for filtering the solutions from the sand-treatment tanks before precipitation.
1 air-compressor, Ingersoll type E, with its receiver, which gives sufficient air to agitate the slime and to run the pump which circulates the solution throughout the mill.
2 sheet-steel cone classifiers, 5 ft. in diameter by 4 ft. high, Spitzlutte type.

Precipitation plant.—This is shown at the right in Fig. 33, and includes:

2 precipitation-boxes, 13 ft. long, 3 ft. 3 in. high, and 2 ft. wide, divided into 7 compartments. Made of redwood, with false-bottom screen, and clean-up valve.
2 tanks of redwood 4 ft. in diameter by 3 ft. high, for washing the precipitate and treating it with acid.
1 melting furnace.

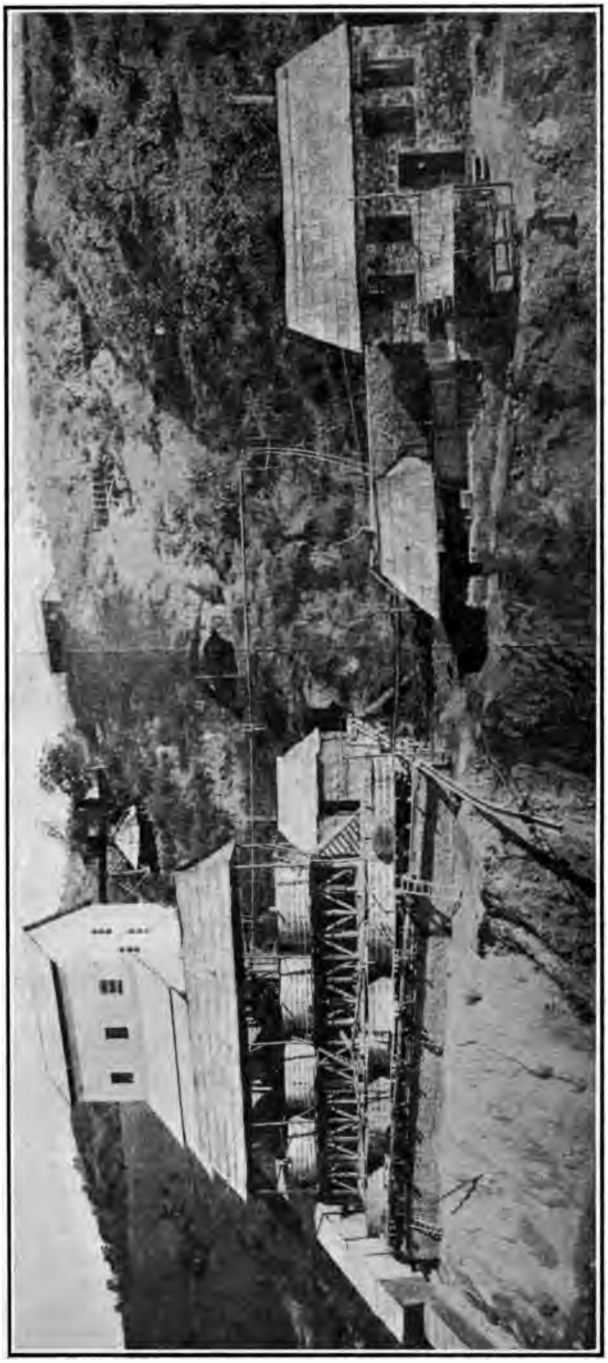


Fig. 33. Mill of the Nuevo Rosario y Anexas Co., Peñoles, Oaxaca.

1 masonry sump-tank 4 ft. by 3 ft. by 4 in. by 4 ft., for clean-up of precipitation-room.
2 redwood sump-tanks 16 ft. in diam. by 6 ft. high, for receiving the strong and weak solutions which have passed through the zinc-boxes.
1 Cameron pump, type 5B, for pumping solution from the sump-tanks to the storage-tanks.

The treatment given in these tanks is about 64 hours and this gives an extraction of about 90% of the gold.

General Remarks Regarding Small Plants.

The number of tanks required in any small mill depends entirely on the quantity of ore to be treated and the time necessary to allow for treatment, in order to obtain a satisfactory extraction. The details given in the description of the methods used in the old mill of the Esperanza Mining Co. will be found useful both in construction and in manipulation, where a small plant of this kind is erected.

The following description of a very inexpensive cyanide plant appeared in the *South African Mining Journal* of September 4, 1897:

The conditions were rather peculiar, as the lot of tailing to be leached was from an arrastra and extremely slimy, fully two-thirds being clean slime. As any heavy pressure would pack such material, the vats were built very shallow, being 2 ft. deep and 12 ft. square. The first solution was added slowly from the bottom, and care having been used in putting the material into the vats lightly, without tamping, there was but little tendency to settle as long as the surface of the solution or wash-water did not fall below the level of the top of the tailing in the vat.

The small amount of material in sight to be handled would not justify the purchase of expensive tanks, especially since an exceptionally large amount of leaching surface was required, owing to the shallowness of the leaching and its extreme slowness in this slimy material. A cheaper method of building tanks had, therefore, to be found; and, after a few experiments, the following was adopted: A row of seven boxes 2 ft. deep and 12 ft. square, braced from the outside every 4 ft., was built, surfaced lumber, 1½ in. stuff, being used throughout, but no extra precaution being taken in the construction beyond thorough nailing. Over the cracks strips of canvas 3 in. wide were laid on top of a streak of hot (but not boiled) asphalt. A sheet of canvas of medium weight, 16 ft. square, sewed with double seams, was then laid down in the vat and folded up in the corners, the sides next the board being covered with a good layer of hot asphalt just as it was laid, causing it to stick fast to the wood throughout. The upper edges were tacked to the vat, no other tacks being put through the canvas. The inside of

the canvas lining was then thoroughly asphalted, with special care about the corners. The vat floor inclined 1½ in. in 12 ft., both east and north. In the northwest corner, 1 ft. from sides, the floor had been cut ¾ of an inch deep in a saucer-shaped depression, and a hole for a 1¼ in. pipe bored. This was put in through a hole in the canvas and bolted tight in hot asphalt with 3 in. washers on the canvas and on the plank below.

Filters were of double burlap on 1 in. square strips, 1 in. apart, laid diagonally on the canvas bottom to within six inches of the sides—these held in place by canvas strips tacked across; four strips 1 by 4 by 11 ft. 6 in.—cut at 45° angle on corners, were laid around the bottom of the vat—a small roll of burlap being tacked to the thin underside, which stuck to the still warm asphalt. To these strips was fastened the burlap filter.

The outlet pipe was bent at right angles 3 in. below the vat. A short piece of hose was wired on it, and into the end of the hose was pushed a wooden spigot, which drained directly into a 2 in. by 4 in. launder running alongside the vats to the zinc-house. This launder was lined with a strip of asphalted canvas, and covered tightly except at the spigots. The same method of preventing leakage by means of asphalt and canvas was used in the construction of zinc-boxes and solution-tanks, the latter being built on the same general plan as the vats.

The pumping from sump-tank to old-solution tank was done by a small sized Pelton motor, so that the waste water from it could be used in leaching. Solutions were carried to the vats from the tanks by means of rubber hose.

So much for the construction, which was certainly cheap; and I will warrant that it was also efficient. With proper care, in the use of well-seasoned timber, and good grade of asphalt, these vats last very well. A leak is easily mended by a new coat of asphalt, and this should be applied every four or six weeks as a precautionary measure. The tanks and vats have given excellent service, and are now being used to leach tailing from a 5-stamp mill since built on the ground.

W. F. Boericke and B. L. Eastman have described (*Mining and Scientific Press,* November 21, 1908) a small 'home-made' cyanidation plant erected by men treating tailing at Grass Valley, California. Such simple plants may be easily erected in out-of-the-way places and at small expense.

CHAPTER XV.

CONTINUOUS CYANIDE TREATMENT.

In Connection With Pachuca Tanks.

In the ordinary process of cyanidation in Pachuca tanks it is customary to employ several, one of which is filling, another emptying to receive the subsequent charge, while the others are undergoing agitation. Consequently there is a loss of time in the use of two of the tanks which might be more profitably employed in agitation. In order to remedy this, and other defects of the present treatment, J. Leslie Mennell and A. Grothe devised a system of continuous cyanide treatment, which is described in *The Mexican Mining Journal* of February, 1909, as follows:

The well known Pachuca tanks are connected up and arranged to operate in series, the first receiving the pulp from the mill, and the last after the passage of the pulp through each tank of the series in each of which it was continually treated, discharging same continuously and by gravity, to the filter plant. In this system of continuous treatment the series or battery of tanks placed at the same level, and consequently the level of the pulp in each tank is approximately the same, there being, however, a gradual slight drop from the first to the last. The level of the pulp in the last tank arranges itself only that slight amount lower than the level in the first tank which enables the pulp to flow from the first to the last tank at the same rate as it is being charged into the first tank. There is thus no handling whatsoever required, as the finished product discharges from the last tank at approximately the level of the intake, and thence proceeds by gravity to the filter plant. It is clear that in this manner all the time now virtually lost in the filling and emptying of each of the tanks is saved, and all the tanks are thus constantly and also usefully employed in cyanide agitation.

The actual treatment which the ore thus receives in passing through the series of tanks is equally as good as that now obtained by separate handling of each tank charge. The number of tanks required depends upon the time of cyanide agitation treatment which the ore needs to secure an economic extraction, but for all ordinary ores requiring up to 48 hours, a series or battery of 4 tanks is sufficient. In a battery of tanks in which they are receiving a

daily quantity of pulp equal to the capacity of each tank, the time of treatment which the ore receives in each tank as it progresses through the series is an average of 12 hours, or in a series of 4 such tanks 48 hours in total. If a longer or shorter time of treatment is required a larger or smaller number of tanks is used.

In the experiments which were made the difficulty which first presented itself was that an ordinary overflow pipe from one tank to another was insufficient, as thereby a concentration of the coarser particles took place in the first tank, diminishing gradually towards the last in the series. To obviate this difficulty a radial collector of the pulp was devised and placed at some distance under the surface in each tank, and in this manner by, as it were, 'sampling' the falling particles in each tank, a uniform transfer of pulp of its average consistency was obtained. That this contrivance answers its purpose is seen from the following sizing tests:

Pulp as supplied to first tank %		Second tank %	Third tank %	Fourth tank %	Overflow %
Under 100-mesh	0.0	0.0	0.0	0.0	0.0
100 to 120-mesh	16.5	16.7	20.9	14.2	18.3
120 to 150-mesh	5.2	6.8	3.6	11.2	6.6
150 to 200-mesh	8.2	9.5	9.5	7.1	6.6
Over 200-mesh	69.5	66.0	64.3	66.6	67.6
	99.4	99.0	99.8	99.3	99.1

The difference between the sizing of the product as charged and as discharged will be seen to be very small and quite insignificant, and points to the conclusion that the ore passed uniformly from tank to tank.

To determine the efficiency of treatment obtained by this continuous system of working in comparison with the older method, tests were made by both systems upon the same ore with all other conditions equal. The fineness of the ore used can be seen from the sizing test indicated in the first column of the above, the strength of solution used in each test was 0.39% and the proportion of solution to dry slime was 100 to 70 in the test by each system. The ore chosen for the test was a very refractory one, requiring 65 to 70 hours agitation in order to extract about 80% of its value (silver). This enabled a comparison of the efficiency of the two systems of treatment when agitated for only 48 hours to be made more accurately, as in neither case was the treatment perfectly finished. The silver was in the state of sulphide, the more refractory being associated with slimed galena and iron pyrite which had escaped concentration, these with iron oxides and a little blende causing considerable cyanide consumption. The test on this ore

by the ordinary system agitated for 48 hours gave the following results:

OLD INTERMITTENT PROCESS.

Amount of fine crushed ore used kg.	Amount solution kg.	Time agitated hours	Strength cyanide solution %	Consumption cyanide kg. per ton	Heads value silver gm. per ton	Residues value silver gm. per ton	Extraction %
70	100	0	.39	0	349	...	....
...	...	12	.26	1.86	...	118	66
...	...	24	.20	2.71	...	108	69
...	...	36	.13	3.76	...	98	71.2
...	...	48	.12	3.86	...	95	72.7

This test was made in a Pachuca tank, an exact miniature in every dimension of the 70-ton tanks in practice; and the test was conducted exactly as in practice.

The test then made upon this same ore by the continuous process was made using 4 of these same tanks connected in series, as above described, with the arrangement for conducting the pulp freely by its own flow from the first to the second, second to the third, third to the fourth and the fourth to discharge. The pulp was fed into the first tank at the rate of 70 kg. per day, with 100 kg. of solution, such being the exact capacity of each separate tank. The first tank was filled to begin with, with ore and solution in the above proportion to save time, and the other three tanks filled with solution for the same reason. The operation of constantly feeding in ore and solution at the rate of 70 kg. of slime with 100 kg. of solution into the first tank of the battery was continued for 8 days. After 5 days, when the process was running perfectly uniformly and satisfactorily, samples were taken from each tank, numbers one to four, and assayed for silver and tested for cyanide strength. The samples were taken frequently from then on up to the eighth day, and the assay value of each determined. The assay value of the samples from each tank were found to give practically constant values, and these values are given below in the tabulated results. After the eighth day solution alone was run in for another 4 days at the same rate, and then when practically all the ore had been discharged from the last tank, the whole of the residues from the test were bulked together in one tank and a sample taken to check up the discharge samples from the fourth tank during the operation of the process. The assay value of these residues was found to exactly check the discharge samples taken while the process was in operation, thus confirming the latter.

The whole test is shown in the table opposite:

NEW CONTINUOUS SYSTEM.

Amount of fine crush-ore fed daily	Amount solution fed daily	No. of tank	Time of treatm't corresponding hours	Strength KCN solution %	KCN kg. per ton	Heads value silver gm. per ton	Residues value silver gm. per ton	Extraction %
70 kg.	100 kg.	......	0	0.39	0.00	352	....	....
....	....	1	12	0.30	1.29	....	146	58.0
....	....	2	24	0.26	1.86	....	122	68.0
....	....	3	36	0.20	2.71	....	101	71.3
....	....	4	48	0.18	3.00	....	80	77.3
Assay of entire discharged residues.......					....	....	80	77.3

These tests demonstrate clearly, as will be seen by the comparison of the figures of the above results, that the continuous process proposed using a battery of Pachuca tanks connected in series, with the radial collecting device mentioned for selecting and insuring a uniform flow of pulp of an average consistency the same as when charged into the first tank, gives results equally as good as are obtained by the present intermittent system of working; in fact, these tests show a slight improvement in results by the continuous system.

The theory of this continuous treatment is as follows: If the tonnage of ore and solution fed continuously into the first tank every 24 hours be equal to the capacity of the tank, then the average time of treatment which the pulp discharging from this tank constantly receives is 12 hours, 50% receiving less and 50% more than this time. The pulp issuing from the second tank constantly receiving its charge from the first tank receives similarly another 12 hours or a total average treatment of 24 hours, only 25% having now received less than 24 hours and likewise 25% having had more than 24 hours. The pulp issuing from the third tank similarly has received another 12 hours or a total of 36 of average treatment, only 12.5% now having had less than 36 hours, while 12.5% has had more than this time.

Upon ordinary docile ores a battery of three tanks is sufficient to enable an economic extraction to be obtained, for with only 12.5% of the ore discharging from the third tank with less than the desired or average time of treatment it is of small practical importance, for this 12.5% has received sufficient time to enable a fairly good extraction to be obtained from it under any circumstances, which extraction also is improved by averaging with it the better extraction got from the 12.5% which has received more than the average time of treatment. However, if a fourth tank be used in the battery, as in most cases will undoubtedly be the practice, the average time of treatment received by the discharging pulp will be 48 hours, and only 6.25% of the ore will have received less than this average time, with also 6.25% having received more. In a fifth tank if one be used in the same battery, the proportion of the pulp

discharging from it with less than the average time will be only 3.125% and from a sixth tank 1.5%, and so on.

It will be seen that a battery of 4 tanks connected in series gives quite good enough results, no more being necessary for the perfect application of the process, the amount of ore issuing from the fourth tank, which has not received as much as the average time of treatment of 48 hours being negligible.

The immense advantages derived from the use of this continuous process in practice will be readily seen. Briefly, they may be tabulated as follows:

1. Avoidance completely of all loss of time for filling and emptying the tanks, which are in consequence always in useful operation agitating the ore with cyanide solution.
2. Elimination of the expense of discharging the tanks, and also that of attention in filling, decanting with wear and tear upon the tanks, etc., incidental to the present intermittent practice.
3. Minimizing of the skilled supervision required in the cyanide treatment, for all the battery of tanks work automatically and together, one man alone whose task will simply be that of a watchman apart from the cyanide chemist being required to work the battery. Each tank is in constant agitation by means of compressed air forced into its air-lift, and the feed by gravity into the first tank, together with the discharge by gravity from the last are the only operations requiring to be looked after.
4. Such a battery of tanks worked in continuous system affords considerable elasticity to the milling operations, not now possible except where spare tanks are installed. If the mill should be stopped for a few hours through accident, it can, upon starting up, be made to put through enough more ore to make up for the loss of time, and the cyanide battery, which would not stop, would take it without discharging residues of appreciably higher assay value.
5. The level of the discharge from the last tank is practically at the same as that of the feed to the first, near the top of the tank; and this saves the considerable height of the tank, and saves pumping the pulp, which runs by gravity to the filter plant. The battery of tanks may in fact be installed in a pit, sunk below the level of the rest of the plant.

Many minor advantages will occur to practical cyanide men, but the first two are the most important which are obtained by the use of the above described continuous cyanide process. A continuous automatic process always commends itself by the simplicity and economy which it introduces, and the above tests show that the system described does not do this at the expense of extraction, but rather improves the latter.

In comment on this description it may be suggested that there are advantages in the ordinary intermittent process over the continuous process which the authors of the preceding article have overlooked. The possibility of keeping a charge, which for any

reason has not yielded a satisfactory extraction, in a tank as long as may be necessary to secure good results; and knowing the exact results which are obtained from each charge before discharging it, is one. This is especially important in a custom mill or a mill where the character of the ore is constantly changing, requiring a different treatment for each batch of ore. In a mill where the ore is all of the same class the continuous treatment may give satisfactory results.

Since writing the above the continuous process has been installed in the new mill of the Esperanza Mining Co., and the results obtained there are entirely satisfactory.

Following the work of A. Grothe, M. H. Kuryla introduced the continuous process at the Esperanza. He has described the results as follows*:

The agitation equipment of the new cyanide plant of the Esperanza Mining Co., El Oro, Mexico, consists of two batteries of six 14-ft. 10-in. by 44-ft. 8½-in. Pachuca tanks. The first battery of six, locally referred to as the 'upper Pachucas,' treats the flocculent slime produced from the coarse crushing by 120 stamps, and is worked on the 'intermittent' or 'cycle' system originally installed. The position permits of a gravity discharge to the storage tanks for the Merrill filter-presses. The six 'lower Pachucas' treat the granular slime resulting from the regrinding of the battery sands in ten 4 by 20-ft. tube-mills. It is with these 'lower Pachucas' this article deals.

Each tank held 80 tons dry slime and 144 tons solution (S:L-1:1.8), and with a feed of 300 dry tons of slime per 24 hours, the time required to fill one tank was 6.4 hours. After the available length of agitation each tank was discharged through a 5-in. valve at the bottom and pulp raised to the filter-press storage tanks by means of three vertical bucket elevators. These elevators were made of an 18-in. 10-ply rubber elevator belt, 54 ft. centre to centre of head and boot pulleys, with 16 in. by 7 in. by 5½ in., type AA malleable iron buckets, spaced 18 ft. centre to centre. The boots were made of cast iron and provided with accessible hand-holes for cleaning. The housing, 2½ by 5½ ft. inside dimensions, was built of No. 12 steel, with removable doors for repairs on belts, buckets, and pulleys. The original belt speed of 360 ft. per min. was raised to 540 ft. per min., to permit one elevator to be held in reserve. With the average condition of the elevators the time required to discharge one tank was about 3.8 hours.

On the assumption that one-half of the filling and discharging times resulted in net agitation on the entire tank charge, the net time lost in these operations amounted to 5.1 hours. The net agitation per charge, accordingly, worked out as follows:

*Inform y Mem del Inst. Mexicano. de Minas y Metalurgia.

	Hours
Total possible agitation without filling or discharging loss...	38.4
Loss in filling and discharging	5.1
Net agitation	33.3
Percentage of net agitation lost in filling and discharging	15.3

The yearly expense connected with the operation of the three elevators figured out as follows:

Three elevator belts, each 120 ft	₱2,160
Power	1,825
Regular attendance	730
Buckets and Bolts	35
Repairs—labor	1,095
Repairs—material (exclusive of belts)	500
Total per year	₱6,345

The difficulties and expense experienced with the elevators made it highly desirable to deliver the pulp by gravity to the filter-press storage tanks (collars of the storage tanks being 6 ft. below the collars of the Pachuca tanks, and 50 ft. distant). Six-inch pipe connections were put in February 10, 1910, first as an experiment to demonstrate the applicability of the scheme. The particular information desired was with reference to comparative extraction and classification during agitation, as it was thought that the 'sandy' product fed to the tanks might undergo considerable classification in passing from tank to tank. The temporary connections proved so satisfactory that they have been retained in present daily operation.

The pulp is fed into the top of tank No. 1, at the rate of 300 tons dry slime and 540 tons solution per 24 hours and flows through the 6-in. pipes by gravity to the discharge box near the top of tank No. 6. The friction drop from tank to tank is about 6 in., making a total drop in the level of the pulp between tank No. 1 and tank No. 6 of 30 in. However, by utilizing the central tube as an air-lift, and so placing the outlet into the final discharge pipe or launder, that the overflow of the air-lift will drop into the outlet, the difference in the level of the pulp in tank No. 1 and the level of the final discharge is reduced to 12 in. In fact, were it desirable to have the final discharge at a higher level than the first tank, the air-lift in the last tank of the series could be raised sufficiently to permit of this being done (at an additional consumption of air, of course). The intake into a tank being near the bottom, the pulp rises through the air-lift, as the current in the annular space around the air-lift is in a downward direction. The discharge from the tank is placed 7 ft. below the collar of the tank and 2 ft. from the side of the air-lift, in this way 'sampling' the overflow of the air-lift, and producing a practically uniform transfer of pulp from tank to tank as shown by sizing tests and solid to liquid ratios in the table opposite:

	Charge	Discharge from tank No. I	II	III	IV	V	VI
Sizing test of pulp:				per cent			
+200 mesh	14.7	14.9	15.2	15.7	15.3	15.1	14.9
—200 mesh	85.3	85.1	84.8	84.3	84.7	84.9	85.1

Strength of KCN solution. Per cent:

0.12	0.112	0.110	.1075	0.105	0.1025	0.100

Consumption of KCN per ton of ore, 358 grams.

Gold contents and percentage of extraction:

	gr.			per cent			
+200 mesh	9.7	46.0	58.0	61.5	62.0	63.0	63.5
—200 mesh	6.0	68.0	75.5	78.0	80.0	82.0	83.5
General sample	6.7	66.0	72.5	74.5	76.0	77.0	78.0

Silver contents and percentage of extraction:

+200 mesh	75	9.0	16.0	22.0	26.0	28.5	31.0
—200 mesh	83	28.0	39.0	46.0	50.5	53.0	54.0
General sample	80	22.5	34.5	41.5	46.0	48.0	50.0

Feed and final discharge—Solid to liquid ratio, 1:1.80.
Sampling 6 days at one-hour intervals.

The gain of extraction in the continuous system as compared to the intermittent is 1.3% of gold and 1.5% of silver and the saving in cyanide 25 gm. per ton of ore. The saving in operating labor and the increase in length of agitation effected in the 'lower Pachucas' has made it advisable to install the continuous system in the 'upper Pachucas,' even though, as already mentioned, the 'upper Pachucas' discharge by gravity to the filter-press storage tanks.

CHAPTER XVI.

CYANIDATION IN PAN-AMALGAMATION MILLS WITHOUT CHANGE IN THE MACHINERY.

This treatment, known as the Gilmour-Young process, has been practiced with good results at the Santa Francisca mines in Nicaraugua. It consists of treating the ore, ground to slime, with an alkaline cyanide solution, and copper-zinc amalgam in ordinary amalgamation-pans. The gold and silver contained in the ore are dissolved by the cyanide solution, and afterwards precipitated by the copper-zinc amalgam and incorporated in the amalgam. After a six-hour treatment in the pans the slime is run into settlers, where the amalgam is recovered in the ordinary manner and treated as usual by filtering, retorting, and melting the resulting bullion. This process is of great importance to small companies already having a pan-amalgamation plant, without the necessary capital to erect a proper cyanide plant. It should be remarked, however, that, on account of the short time of contact between the ore and the cyanide solution, this treatment is only applicable to slime, as it cannot be expected that sand will yield good extraction. Furthermore, for the same reason, it will be found that in general this treatment will only be applicable to gold ores, as silver ores where the silver will be dissolved in the short time given in this process are very rare. The following description of the process is taken from Alfred James' 'Cyanide Practice':

Cyanide solution is made up containing KCN 0.2% and NaOH 0.1%. The pans (5-ft. Boss) are charged as thickly as is consistent with a good circulation of the pulp, the cyanide solution being run in at the same time as dry ore is charged into the pan, until the charge is sufficiently watered down to circulate well. Two or three bottles of mercury are added, and the pan run without further addition for about two hours, when 30 lb. of amalgam are added, and at the end of another four hours the precipitation is complete, and the pan is discharged into a 7½-ft. settler, in which the mercury separates out. The amalgam is used over and over again with fresh charges of pulp until it becomes rich enough to retort, and by passing the amalgam ten times through the pans a bullion 700 fine is obtained. The precipitation is quite as complete with rich amalgam as with poor, but if the amalgam becomes very rich, the quantity of such rich, and consequently less active, amalgam must be increased. The point in this process where most care must be taken is that the mercury rises well with the pulp. In the 5-ft. pans the

speed must not fall below 68 revolutions per minute. The mullers must be kept well down, but need not be set to grind.

In case of the extraction not being satisfactory, find whether the gold remains in the sandy portion of the ore, 1, or the clay portion, 2, or in solution, 3.

1. If it remains in the sand, the precious metals can only be extracted by re-treatment in a leaching plant, or by re-grinding.

2. If in the clay portion, the want of extraction is due to any of the reasons which affect extraction in the ordinary cyanide process, and may be rectified by experiments in the laboratory.

3. If the gold remains in solution, and the precipitation is not complete, the cause will be found in the faulty circulation of the mercury or faulty precipitation due to one of the following reasons: (**a**) Consistency of the pulp too thin; (**b**) Speed of agitation too slow; (**c**) The mullers of the pan are not kept nearly touching the bottom of the pan.

The copper amalgam can be made in one of the amalgamating-pans, but, owing to the action of the copper sulphate on the iron portions it is better to make it in a special pan. A convenient charge consists of 100 lb. copper sulphate, 30 lb. iron filings or cast iron turnings, and 6 bottles mercury. The reaction is complete in from two to four hours, depending on the condition of the filings. This charge produces about 150 lb. of squeezed amalgam containing 14% to 15% of copper. It is not necessary to use much zinc amalgam with the copper amalgam. With some ores none at all is used. The best way to use the zinc amalgam, made by pouring ten parts of mercury into one part of molten zinc, is to add about 50 lb. of zinc amalgam to 300 lb. of copper amalgam and mix the two well together. The chief advantage of the small proportion of zinc is that it preserves the amalgam from being reduced in weight to the same extent that pure copper amalgam would. Both the zinc amalgam and the zinc-copper amalgam should be kept under water to prevent rapid decomposition. The average ore of the Santa Francisca mine treated by this process contained 1.3 oz. of gold and 3.6 oz. of silver per ton, and the extraction obtained averaged 89.3% of the gold and 75.8% of the silver. The results of one month, however, were higher than the average, being 94.0% of the gold and 82.2% of the silver.

CHAPTER XVII.

PRECIPITATION ON METALLIC ZINC.

Use of Zinc Shavings.

The success obtained in precipitating gold and silver from cyanide solutions is due to Messrs. MacArthur and Forrest, who proposed the use of zinc in very fine threads or shavings. One gram of gold per metric ton of solution represents, by weight, one part in one million and by volume one part in twenty million, so that it is evident that, for each drop of the solution to come in contact with the zinc, the surface of this latter must be extensive. This is obtained by cutting the zinc into shavings. Each kilogram of ordinary zinc-shavings has a surface of 10 square metres, more or less, so that one ton of zinc-shavings represents a surface of about one hectare (2.48 acres). The shavings are cut from 1/32 to ⅛ of an inch wide, and from 1/400 to 1/800 of an inch in thickness, and are placed in a loose mass within the precipitation boxes, so that the solution may pass through them and deposit its gold and silver contents.

The ordinary zinc-box consists of a long box divided by double partitions into five compartments, each about 3 ft. long, 3 ft. wide and 3 ft. deep, and arranged with a false bottom about 4 in. above the bottom of the box, on which the zinc-shavings are placed. The partitions are so arranged that the solution enters into the box below the false bottom, passes upwards through the zinc, overflows above the first partition, and passes under the second partition, to enter into the second compartment below the false bottom.

Manufacture of Zinc-Shavings.—In some mills it is customary to cut the zinc in round discs from 5 in. to 12 in. in diameter, with a 1-in. hole in the centre in order to fix them on a lathe; the cutting being done with a wide carpenter's chisel while the lathe is turning. In other mills, where large quantities of shavings are required they use special lathes for the purpose, or else take an ordinary lathe, make a mandril for it from a round log of wood 3 ft. long and 9 in. in diameter, having a longitudinal slot to receive the edge of the first sheet of zinc, and roll five sheets of No. 9 commercial zinc around it, allowing each sheet to lap about 6 in. over the succeeding one, to keep them tight. When the roll is finished, it should be tightened by several wraps of iron wire, and is then ready for cutting, using a square-headed cutting tool.

The zinc shavings should be removed from the lathe as formed, by means of a pair of suitable rolls, moved by the same machinery. Practice has demonstrated that the zinc should not be cut into shavings thinner than 1/800 to 1/400 of an in., as otherwise they are easily broken into short pieces, which not only fill the interstices between the shavings, forming thus a compact mass which does not allow the free passage of the solution, but also they fall through the screen bottom and mix with the precipitated gold and silver, where they increase the cost of refining or melting, by requiring more acid or flux in that treatment.

Chemical Reactions During Precipitation.—The chemical reactions which occur during precipitation are neither simple nor entirely understood, as it is recognized that there are certain electrochemical or galvanic reactions, which exert an influence on the final reaction. Furthermore the presence of so many complex salts in an ordinary mill solution makes it impossible to express all of the factors which may modify the reactions. However, it is generally agreed that the precipitation of the gold and silver is due to the reducing action of nascent hydrogen, produced by the reaction of the zinc with the cyanide solution, and that this gas precipitates the gold and silver from the double cyanides, in accordance with the following reactions.

$$(a)\ Zn+4K\ CN+2H_2\ O=K_2\ Zn\ (CN)_4+2KOH+2H.$$
$$(b)\ 2\ KAu\ (CN)_2\times 2KOH+2H=2\ Au+4KCN\times 2\ H_2\ O$$
$$(c)\ 2KAg\ (CN)_2+KOH+2H=2\ Ag+4KCN\times 2\ H_2\ O.$$

As the reaction (*a*) takes place to a greater extent than the reactions (*b*) and (*c*) it causes a certain amount of caustic alkali always to be present, or in the act of formation, in the zinc boxes. This alkali, in the presence of oxygen, may dissolve the zinc forming potassium zincate and hydrogen peroxide, according to the following reaction:

$$(d)\ Zn+2\ KOH+O_2=K_2\ Zn\ O_2+H_2\ O_2.$$

The hydrogen peroxide thus formed has a pernicious action in the zinc boxes, as it may cause the re-dissolution of the gold and silver, thus causing the solution to leave the boxes richer than they were when entering; fortunately however, the quantity of oxygen present in the solution is small, and the presence of lime also hinders such action, as it destroys the peroxide.

$$(e)\ Ca\ (OH)_2+H_2\ O_2=CaO_2+2H_2\ O.$$

The calcium dioxide thus formed is insoluble in alkaline solutions, and falls to the bottom of the boxes. The caustic alkali formed according to the reaction (*a*), may react with various substances present in solution or it may dissolve silica, and hydroxide or ferrocyanide of zinc which might have formed coatings on the surface of the zinc

shavings. The reactions (*a*), (*b*) and (*c*) express the conditions which are present when treating simple cyanide solutions. As the ordinary mill solution contains, in addition to the simple cyanides, all sorts of double cyanides (zinc and lime, zinc and potassium, potassium and lime, etc.), whose composition varies according to the alkalinity of the solution, ferrocyanides, sulphocyanides, and numerous complex cyanogen compounds, it can readily be understood that it is impossible to state exactly the reactions which actually occur in the zinc boxes. However, it may be considered that the reactions (*a*), (*b*) and (*c*) represent the general type of the conditions which govern zinc precipitation, for the purpose of calculations referring thereto.

Causes and Remedies of Poor Precipitation.

The causes of poor precipitation may be physical or chemical. Among the physical causes are:

Small Surface of Zinc.—The surface of the zinc-shavings may be insufficient for the precipitation of the quantity of solution passing over it in a given time, either on account of the shavings being too thick, or on account of not having enough compartments. In order to eliminate this factor from the possible causes it is well to arrange matters from the start; that the zinc used is cut in shavings from 1/32 to 1/8 of an in. wide, and from 1/800 to 1/400 of an in. in thickness; and have the zinc-boxes of such a capacity that they contain at least 1 cu. ft. of zinc-shavings for every 2½ tons of solution to be precipitated daily. The practice in this respect varies in different mills, as will be seen in subsequent chapters treating of cyanide practice, but it has been found in the mill of the Esperanza Co., in El Oro, that a satisfactory precipitation may be obtained with this capacity, although the zinc-boxes in the El Oro and Dos Estrellas mills have two, three and five times this capacity, it being considered better to have too large rather than too small a capacity.

Too Great Velocity of Current.—The current may pass through the zinc-boxes so rapidly that it does not give time for complete precipitation. The velocity of the current depends in part on the width and capacity of the zinc-boxes, as explained in the last paragraph. The proper velocity varying between 0.725 and 2.5 tons of solution per cubic foot of zinc, per 24 hours. It also depends on the shavings being loosely arranged, permitting free passage through the interstices, as hereafter explained. One ton of solution per cubic foot of ordinary zinc-shavings, per 24 hours represents a contact of 45 minutes; but at times, when working with very weak solutions, this time of contact is not sufficient, especially when the zinc is covered with a coating of any substance which prevents direct contact, and in such cases the solution must flow less rapidly.

Zinc Agglomerated in a Compact Mass.—When the zinc-shavings are agglomerated in a compact mass, due to the presence of 'shorts,' to precipitated gold and silver, to slimes not filtered from

the solution before precipitation, or to any chemical precipitate which may fill the interstices between the shavings, the solution, not being able to pass through the shavings, finds a channel at the sides of the box, or through some hole in the agglomerated mass, and, as it consequently does not come in contact with all of the zinc, but only with a small portion of it, the result is an imperfect precipitation of the values contained in the solution. If the agglomeration is due to 'shorts' present in large quantities, they should be separated from the longer threads or shavings of zinc, and placed in trays with screen bottom, one above another, using No. 10 screen, with a layer of about 2 in. of 'shorts' in each tray, in a separate compartment of the zinc-boxes. If due to precipitated gold and silver, this should be removed by shaking the shavings from time to time to allow the precipitate to fall to the bottom of the box. If due to slime, which has not been separated from the solution before passing it through the zinc-boxes, this may be remedied by passing the solution through sand-filters previous to entering the zinc-boxes. If due to the fact that there are currents through certain parts of the shavings, owing to the shavings being of unequal thickness, which produces interstices larger than in other parts, this may be remedied by suppressing the cause through using shavings of uniform size. If due to the formation of any chemical precipitate, it may be remedied by preventing its formation, as will be explained subsequently.

Slime Covering the Surface of the Zinc.—When the solution entering the zinc-boxes contains slime which may coat the zinc with a film that would prevent contact between it and the solution, the remedy, as already explained, is to filter the solution.

Among the chemical causes of poor precipitation are:

Zinc Too Pure.—Chemically pure metallic zinc has very little effect as a precipitant for gold and silver, as the presence of a galvanic couple is necessary to induce precipitation. This galvanic couple may be formed by the metallic zinc with any other metallic element which exists as an alloy in commercial zinc; or with the metallic gold and silver which may be precipitated on the zinc; or a zinc-lead couple may be formed by immersing the zinc shavings in a bath of lead acetate, whereby a deposit of metallic lead is formed on the surface of the zinc before placing it in the zinc-boxes; or the lead acetate may be added to the cyanide solution entering the zinc-boxes.

When it is observed that the zinc-shavings, in the various compartments of the zinc-box, retain their primitive lustre, it is probable that the gold and silver are not being precipitated; this may be remedied by dipping the zinc in lead acetate solution, or adding the lead acetate to the head of the zinc-boxes.

The Presence of Soluble Sulphides.—These sulphides may attack the double potassium-zinc cyanide forming zinc sulphide which may deposit on the surface of the metallic zinc, preventing the precipitation of gold and silver, as the impermeable film of zinc sulphide prevents contact with the solution. This precipitated sulphide causes greater damage in weak than in strong solutions, as these latter may partly dissolve it. The remedy for soluble sulphides is to add lead acetate to the treatment tanks, in order that any soluble sulphides may be there precipitated and removed from the solution before it enters the zinc boxes.

Solution Not Sufficiently Alkaline.—When the solution is not sufficiently alkaline, and does not contain the amount of lime necessary to precipitate, as calcium sulphate, the sulphuric acid, derived from the oxidation of the pyrite, which it may contain; this acid instead of forming insoluble calcium sulphate, may form soluble calcium bisulphate, which would remain in solution until it arrived at the zinc-boxes, where, upon meeting the caustic alkali which is generated in the zinc-boxes in accordance with reaction (*a*) previously explained in this chapter, one part of the acid sulphate would unite with this alkali, converting the bisulphate into normal calcium sulphate, which, being insoluble, might form such a volume of precipitate that it would entirely fill the interstices between the zinc-shavings, and cover them to such an extent that the precipitation of gold and silver might be entirely prevented. Also in case lime is not used, if the quantity of caustic soda or potash used to obtain the required alkalinity is not sufficient, a bisulphate may be formed, which, although it would not cause the precipitate heretofore mentioned, would make the solution so acid, that it would cause the destruction of the cyanide, and interfere with the precipitation.

Solution Too Alkaline.—If the alkalinity of the solution is due to lime it has very little effect on the precipitation, up to a certain point; but if the alkalinity is due to caustic soda or potash, it may cause considerable trouble in the zinc-boxes, not only by the formation of hydrogen peroxide which may redissolve the gold and silver already precipitated; thus making the solutions leaving the zinc-boxes richer than when entering, but also by increasing the consumption of zinc, since, as has already been explained, these alkalis attack and dissolve this metal. The remedy to be applied for an excess of alkalinity, when due to caustic soda or potash, is to add a quantity of sodium bicarbonate or bisulphate, or a dilute solution of sulphuric acid to the solution entering the zinc-boxes; but for reasons explained in paragraph (7), this remedy may not be applied in the presence of alkalinity due to calcium hydrate or lime, since it might form the precipitate of calcium sulphate previously mentioned.

It is much better to take pains in the beginning to not make the solution too alkaline, and to use lime instead of caustic soda or potash for neutralizing the acidity of the ore. Should the solution be too alkaline, due to the presence of calcium hydrate, this may be remedied by adding sodium bicarbonate to the solution in the treatment tanks, so that the lime may be precipitated, and discharged with the tailing as calcium carbonate, and not enter into the zinc-boxes. It is only rarely that this remedy has to be applied, as the majority of ores contain acidity which will attack the lime, so that by simply refraining from adding lime to the ore, within a short period any excess of alkalinity would be destroyed.

Beneficial Action of Lime.—In addition to the action of lime in neutralizing the acidity of the ores, in coagulating the slimes, etc., it acts in another manner, which is of great benefit in cyanidation, in precipitating the zinc from cyanide solutions, thus maintaining the quantity contained as double potassium-zinc cyanide at a more or less fixed point. The chemical reactions which occur when gold (or silver) are dissolved by double potassium-zinc cyanide in the presence of lime have been expressed by W. H. Virgoe in the following equation:

$$(f)\ 2\,Au + K_2Zn\,(CN)_4 + O + H_2O + 3\,Ca\,(OH)_2 = 2KAu(CN)_2 + K_2Zn\,O_2 + CaZnO_2 + 2\,Ca\,(CN)_2 + 4H_2O.$$

The zincates of potassium and calcium are soluble in the solution so long as it contains an excess of potassium cyanide; but upon meeting any carbonic acid, by exposure to the air or otherwise, the zinc is precipitated as zinc carbonate, or as double zinc-calcium carbonate; also upon meeting any soluble sulphides in solution, the zinc is precipitated as zinc sulphide, so that in one way or another the zinc is thrown out of the solution.

The double potassium-gold cyanide, formed according to reaction (*f*) deposits its gold content when passing through the zinc-boxes, with the formation of double potassium-zinc cyanide (reactions (*a*) and (*b*)) and the calcium cyanide formed according to reaction (*f*) has the same solvent power as potassium cyanide in proportion to its cyanogen content, so that due to the presence of lime in the solution there is a constant cycle of formation and decomposition of the double potassium-zinc cyanide. G. W. Williams, in volume IV of the Proceedings of the Chemical and Metallurgical Society of South Africa, states that potassium cyanide will attack calcium carbonate (which is a frequent constituent of gold and silver ores), forming double potassium-calcium cyanide and potassium carbonate, and the potassium carbonate so formed may react with the zincates in solution, precipitating the double zinc-calcium carbonate which is insoluble and will be discharged with the tailings. Zinc may also be precipitated from its solution upon meeting any ferrocyanides in the solution, but it is probable that the greater proportion of the displacement of the zinc is due to the presence of lime in the solution.

Some authors claim that an excess of lime causes a large consumption of zinc; but it should be understood that this consumption is not directly caused by the calcium hydrate but rather by other solvents such as potassium cyanide or potassium hydrate which are liberated by the calcium hydrate. The reaction (*a*) explains how zinc is dissolved by potassium cyanide, with which it enters into combination forming the double potassium-zinc cyanide, and reaction (*f*) explains the decomposition of this double cyanide due to the presence of lime. Calcium hydrate also may decompose potassium cyanide, forming calcium cyanide and potassium hydrate, according to the following reaction:

$$(g)\ 2KCN + Ca\ (OH)_2 = Ca(CN)_2 + 2KOH.$$

and the potassium hydrate thus formed may consume zinc, according to reaction (*d*). Calcium hydrate of itself has a very slight solvent action on zinc, but as the potassium and sodium hydrates, formed as in the preceding reactions, are energetic solvents of zinc, it is considered that lime or calcium hydrate is indirectly a solvent of zinc.

From the preceding statements it is evident that a large excess of lime may cause a heavy zinc consumption, so that, although there should always be an excess of lime, or rather an excess of alkalinity due to lime, this alkalinity should not be greater than necessary to fulfill these conditions:

I. It should be sufficient to coagulate and settle the slime, and produce clear solutions.

II. It should be sufficient to neutralize the acidity of the ore.

III. It should be sufficient to regenerate the greatest possible quantity of free cyanide.

This last condition is obtained when there are two per cent of lime for every per cent of zinc in the solution; but it may be easily determined exactly, by analyzing the solution for free cyanide, and then adding decinormal solution of calcium hydrate (or sodium hydrate) gradually to the mill solution analyzing the mill solution for free cyanide, after each addition of alkali, until it is found that a further addition of alkali does not increase the free cyanide in solution. The quantity of alkali necessary for this purpose, is the minimum that the solution should contain. Should the analysis be made with decinormal solution of caustic soda, the quantity of lime which should be added may be calculated from the result, understanding that 28 kg. of lime have the same alkalinity as 40 kg. of caustic soda.

The Famous White Precipitate.—This precipitate, which is often found in zinc-boxes, covering the shavings, and thus interfering with precipitation, has been analyzed by competent chemists; but the results obtained by these analyses are so unlike in different mills, that the author believes that there are two classes of white precipitate, and that the class formed in any particular case depends on the alkalinity of the solution.

The analyses of these two classes appear in Vol. IV of the Proceedings of the Chemical and Metallurgical Society of South Africa.

The first class, which the author believes to be that most frequently encountered when treating weak solutions, very slightly alkaline, was analyzed by Professor A. Prister, with the following results:

	Per cent
Insoluble matter	6.30
Alumina and iron oxide	3.20
Lead	trace
Zinc ferrocyanide	26.25
Zinc cyanide	17.90
Metallic zinc	28.44
Sulphuric acid	4.50
Calcium oxide	3.50
	90.09

Professor Prister states that the cyanide and ferrocyanide of zinc, being insoluble in dilute sulphuric acid, are the cause of the bulk of the white precipitate found in the zinc-boxes not being reduced by acid treatment. He states also that the weaker the solution the greater the quantity of the white precipitate which will be formed, as both the cyanide and the ferrocyanide of zinc are maintained in solution by an excess of free cyanide or of free alkali.

This first class of white precipitate is properly the 'famous white precipitate' which caused so much difficulty in the precipitation of gold and silver from weak cyanide solutions, and is almost always present when treating solutions contained in mill-ponds formed by the tailing of previous operations. In these ponds the cyanide solution continues dissolving the gold and silver contained in the tailings, and at the same time dissolves some of the metallic iron or iron salts, which are ever present in the tailing, forming potassium ferrocyanide. This ferrocyanide, when passing through the zinc-boxes, reacts with the double potassium-zinc cyanide which is therein formed (equation *a*), and precipitates the ferrocyanide of zinc on the shavings, owing to the fact that the cyanide solution is so weak that it cannot maintain the ferrocyanide of zinc in solution. The cyanide of zinc, and the double potassium-zinc cyanide which are present in the solution of the ponds, are held in solution by the presence of a slight excess of potassium cyanide, but on passing through the zinc-boxes this excess of free cyanide combines with the metallic zinc-shavings, and at the moment when the excess of free cyanide is consumed the zinc cyanide precipitates on the shavings, as it was only kept in solution by the excess of cyanide. The remedy to apply to this class of white precipitate, is to strengthen the solution, adding strong solution to the head of the zinc-boxes, or in case the solution is to be thrown away after precipitation, and it is not desired to throw away the cyanide, the zinc-shavings, which are covered with the white precipitate, may be removed from the boxes and dipped into a special box containing a

2% solution of cyanide or of caustic alkali, in order to dissolve the white precipitate and leave the shavings clean and ready for subsequent use.

The second class of white precipitate, which probably is formed when very weakly alkaline, even less than in the preceding case, consists, according to G. W. Williams, principally of hydrated oxide of zinc, with variable small quantities of calcium carbonate, alumina, and at times as much as 8% ferrocyanide of zinc. A sample of this class of precipitate, analyzed by A. Whitby showed the following composition:

	Per cent
Silica and insoluble	2.10
Zinc oxide	54.00
Iron oxide and alumina	0.75
Lead oxide	0.73
Calcium oxide	0.50
Cyanogen	2.50
Potassium oxide	0.70
Loss on ignition (H_2O, CO_2, etc.)	36.80
	98.08

G. W. Williams explains the solution of gold by the double potassium zinc cyanide according to a slightly different reaction than that of (*f*) proposed by W. H. Virgoe, and he may be correct in this, as according to equation (*g*) there would be no calcium hydrate present in the solution, since it would all have been converted into calcium cyanide, with the evolution of caustic potash. At least his reactions explain the solution of gold by neutral solutions of the double cyanide, and also the formation of the white precipitate which he examined. The reactions according to Williams are:

$$(h)\ 2\,K_2Zn(CN)_4+2\,Au+O+H^2O=2\,KAu(CN)_2+2KOH+2Zn(CN)_2+H^2O.$$

The caustic potash reacts with the zinc cyanide in various proportions, according to the concentration, giving a final result of a partial conversion into potassium-zinc double cyanide, as follows:

$$(i)\ 2\,Zn(CN)_2+(2+x)\,KOH=K_2Zn\,(CN)_4+Zn\,O(K_2O)\frac{X}{2}+\frac{X+2}{2}H_2O,$$

and giving at the same time a precipitate of oxide of zinc and cyanide of zinc, due to the lack of equilibrium in the solution. The presence of caustic soda or caustic potash interferes with the precipitation of the oxide and cyanide of zinc, and when present in excess entirely prevents their precipitation. Consequently, when there is an excess of alkalinity in the solution the conditions would probably approximate those expressed in equation (*f*), and it is also probable that the difference in alkalinity is the cause of the difference in the character of the white precipitate, to a certain extent. The presence of ferrocyanides will naturally depend on their existence in the solution. The remedy to be applied to prevent the formation of the white precipitate is to

make the solution alkaline by the addition of lime to the treatment tanks, or to increase the strength of the cyanide solution in the zinc-boxes.

A third class of white precipitate has been mentioned on page 162, but this is entirely different from the famous white precipitate.

Weak Solution.—When the solution is very weak in cyanide it may cause the formation of the white precipitate, as explained in the last paragraph, which would cover the zinc and thus prevent the precipitation of the gold and silver. The remedy is as previously explained. Also it may cause poor precipitation by passing too fast through the zinc-boxes, as a velocity suitable for a strong solution might be too great for a weak solution. Remedy by passing more slowly. The precipitation from very weak solutions by ordinary zinc shavings is often imperfect or nil, whereas shavings previously dipped in lead acetate may cause a perfect precipitation of the values.

It sometimes occurs, in precipitating weak solutions, that after a day or two the precipitation ceases, or becomes imperfect, even though the precipitation is made by zinc-shavings previously dipped in lead acetate. When this occurs it is probably due to slimes or white precipitate which mechanically cover the zinc and prevent precipitation, and the best remedy to apply is to clean up the boxes twice a week, washing the shavings, if necessary, in a 5% solution of caustic soda before replacing in the boxes.

Gelatinous Coating on Zinc-Shavings.—Such coatings may consist of silica or of alumina. When they consist of silica they may be dissolved by caustic soda, either added to the first compartment, or applied by dipping in a separate tank. When they consist of alumina they may be dissolved in a separate tank, by dipping the shavings in a solution of dilute sulphuric acid. This acid treatment however is seldom recommended, as it is both expensive, and dangerous, as will be explained subsequently under acid treatment.

Presence of Copper in Solution.—When copper exists in cyanide solutions, especially in weak solutions, it will precipitate on the surface of the zinc-shavings in such a smooth, even film that it may entirely cover them, and prevent the contact of the solution with the zinc, thus diminishing the action of the galvanic couple to such an extent that after a short time the precipitation of the gold and silver will entirely cease.

The following remedies have been proposed to overcome this difficulty: (*a*) Increase the strength of the cyanide solution to such an extent that the copper will remain in solution while the gold and silver are precipitated. This remedy has the disadvantage that after a short time the solution will become so charged with copper that it will not have sufficient strength to dissolve the gold and silver, and that it may require the use of a stronger solution than is practicable, on account of the high cyanide consumption, as such solutions would have to be renewed with great frequency. However, there is a certain strength of solution at which the copper precipitates in a flocculent mass without covering the zinc with an impermeable film.

The following method, proposed by W. H. Virgoe, has been practiced successfully at Minas Prietas, Sonora. The solution, after being strengthened to such an extent that it will maintain the copper in solution, is passed through zinc-boxes containing ordinary zinc-shavings, upon which the gold and silver are precipitated. The solution is then slightly diluted, and passed through a second series of zinc-boxes containing the lead-zinc couple, formed by dipping the zinc-shavings in a bath of lead acetate previous to placing in the compartments; here all of the copper is precipitated, as well as any gold and silver which may have remained in solution.

(*b*) Precipitate by means of the lead-zinc couple, instead of by ordinary zinc. This method has given satisfactory results in the Coscotitlan mill. In Pachuca, Hidalgo, where they secure an almost perfect precipitation of the gold and silver from a solution containing 0.05% KCN, and carrying considerable copper. The reason the zinc works better, when covered with lead, in precipitating solutions containing copper, apart from the galvanic action due to difference of potential, is that the surface of the lead coating is rough, having innumerable points or irregularities, at each one of which exist the most favorable conditions for the disassociation of hydrogen gas by the electrolytic action, whereas in a smooth plain surface these points, where the hydrogen may be disengaged, only exist at the edges, and, since the precipitation is due to the hydrogen, when there are less points where it may be disengaged, there will be less gas produced, and consequently less precipitation.

Precipitation by Zinc-dust or Zinc-powder.

This method of precipitation was proposed by H. L. Sulman, and has been installed in several mills in the United States, Canada, and Mexico. As in most new methods, various defects were encountered in the first installations in the Black Hills, South Dakota, so that some of the mills which had installed this process have discarded it and returned to precipitation on zinc-shavings; recent improvements have overcome most of these defects, and it is now claimed that it is superior to precipitation on shavings, both in cost of operation and in consumption of zinc. However, this may be under certain favorable circumstances, it has not yet been so proved in general, and it is improbable that it will ever entirely replace the shavings, as these latter do not require expert attention, and in small works require no expensive installation.

The advantages of precipitation with zinc-fume, are that the zinc in this form has a greater surface than is possible in any other form; that it requires, consequently, less time of contact, in order to precipitate the gold and silver, and on this account should theoretically consume very little zinc. In practice, however, it is found that the consumption is high, probably on account of the fact that, in order to be certain that a sufficient amount of zinc is added, it is necessary to add a considerable excess.

The chief difficulty consists in the recovery of the powder after precipitation. The method of procedure formerly used in precipitating

with zinc-fume, was to mix the solution with the proper quantity of zinc-fume in an agitation tank; agitate with compressed air for five minutes (which time had been demonstrated by practice to be sufficient for complete precipitation); the solution was then filtered, either in filter-presses, or by passing through boxes, similar to the ordinary zinc-boxes, whose compartments are filled with ordinary cotton waste, or preferably with oakum; the collected precipitate was then treated by the ordinary methods.

In modern practice the zinc-dust is introduced directly into the suction of the pump which forces the solution through the precipitate filter-press. In the De Lamar mills in Utah, where the fume precipitate was collected in filter-presses, it was found that the zinc consumptation was from 3 to 4 lb. per ton of ore. Comparing this result with precipitation on shavings, where the consumption varies between 0.3 and 2.6 lb. of zinc per ton of ore treated, it is evident that the advantage is in favor of the older method. However, as heretofore stated, modern improvements, and exact calculations as to the quantity of zinc to be added to each precipitation tank, may improve these results.

In the Esperanza mill in El Oro, state of Mexico, where the Merrill zinc-dust precipitation system has recently been installed, it is claimed that this process is vastly superior to the zinc-box precipitation, in that it is cleaner, as the zinc-dust is fed automatically to the suction of the pump which forces the solution through the Merrill prceipitation press, so that there is no loss of solution or precipitate, such as occurs in the clean up of the precipitation boxes where both solution and precipitate may be dropped on the floor in handling the shavings; that there is less liability of theft, as the precipitates are only handled when taken from the press; that less labor is required, as one man can attend to the whole process of precipitation and filtration; that the precipitation is perfect; that zinc-dust costs less than zinc-shavings; and that less zinc is consumed, as in this mill they formerly consumed 1.64 lb. of zinc per ton of ore, whereas they now require but 1.5 lb. of zinc-dust.

Commercial zinc-dust, according to the analysis of R. H. Harland, contains 1,74% of lead; 0.69% of cadmium; 0.11% of iron; traces of arsenic; 0.19% of silicious matter; 0.52% of carbon, and from 3% to 5% of zinc oxide. Before using, the fume should be treated with a dilute solution of ammonia in order to dissolve the oxide of zinc, after which treatment it is ready for use.

Since this chapter was first written vast improvements have been made in the manner of precipitation by zinc dust, by means of automatic mixers and filter presses, so that the consumption of zinc has been materially lessened. Consequently, due to the cleanliness of the method, the small amount of labor required, and the possibility of locking up the presses so that thieving is made difficult, the majority of the mills now being erected are installing this method of precipitation. Details of the practice in zinc-dust precipitation will be found in the latter part of Chapters VI and XII.

CHAPTER XVIII.

TREATMENT OF CYANIDE PRECIPITATES.

Clean-up.

The method employed in cleaning up the zinc-boxes varies according to the manner in which the precipitates are subsequently collected, refined and melted. In mills where the precipitates are given an acid treatment before melting, it is customary to let clean water run through the zinc-boxes for an hour before beginning the clean-up, not only to avoid loss of cyanide solution, but also to avoid the formation of hydrocyanic acid during the acid treatment. In modern practice in Mexico, in the majority of mills, as the acid treatment is not used, this water wash is not required, as there is no formation of this dangerous gas in direct melting, and the loss in cyanide due to direct melting of the slime which contains solution, is not greater than the loss which would be caused by dilution of the liquid.

In some mills, before the clean-up, they run very strong cyanide solution through the zinc-boxes, in order to loosen the precipitate from the shavings. Such practice is quite expensive in zinc consumption, and, as any precipitate not collected in one clean-up will be obtained in the next, is unnecessary. In some mills it is customary to add a solution of alum, or a little sulphuric acid, to the precipitated slime, after taking them from the zinc-boxes, in order to be able to decant the clear liquid more easily, after the slime has settled. In modern mills the use of filter-presses makes a more complete separation of the precipitate from the solution, with less loss of time and without requiring chemicals to produce settlement.

The different methods practiced in treating precipitates derive their origin from attempts to overcome the difficulties caused by the presence of zinc 'shorts' in the precipitate, which formerly interfered with precipitation by filling up the interstices between the shavings, thus forming a compact mass impermeable to the solution, so that some method had to be devised for treating the shorts. The modern Mexican practice of placing the shorts in separate trays in a special compartment has overcome this difficulty to a great extent, as the shorts are thus almost entirely consumed in their proper work of precipitating and the zinc consumption is less, as they rarely have to be destroyed either by direct smelting or acid treatment.

Details of Operations in a Clean-up.—Various methods are employed in making a clean-up of the zinc-boxes, but the follow-

ing details of the practice in the San Francisco mills in Pachuca will give a fair idea of the ordinary Mexican practice. The valve admitting solution into the first compartment of the zinc-box is closed, and the clean-up begun in this compartment. Using rubber gloves, the top layer of zinc-shavings are shaken and washed in the solution of the same compartment until the gold and silver precipitate, and zinc shorts, which were mixed with the longer threads of the shavings, have been separated from them. The long clean shavings are then lifted out of the first compartment and placed on a tray having a screen bottom, which rests over the second compartment. Continue cleaning and removing the top layer of shavings, until there is a depth of about eight inches of solution above them. Then partly open the valve in the bottom of the first compartment, so that the solution may gradually sink in the compartment, at such a rate that it keeps enough solution above the shavings for washing them as they are being removed. When all of the long shavings have been removed, down to the false bottom which supports them, the discharge valve in the bottom of the compartment is entirely opened, and, using a ¾-in. hose connected with a solution pipe, the sides and false bottom are thoroughly washed with solution. The false bottom is then removed, and washed, and any precipitates and 'shorts' which remain in the bottom of the compartment, are washed through the discharge valve until the bottom is clean. The valve is then closed; the false bottom replaced; the washed shavings then rearranged in the compartment in even, loose layers; and the solution valve is again opened until the first compartment is filled with solution, when it is closed, to allow the second compartment to be cleaned.

The second compartment is cleaned in the same manner, using cleaned shavings from the second compartment to replace what may be lacking in the first, and so on to the last compartment, which is filled up with new zinc. All solution, precipitates, and 'shorts' which pass through the valve in the bottom of the compartments, fall into a trough which conducts them to a box, having a bottom of No. 20 iron-screen cloth, placed over a small tank, so that, while the solution and precipitated slime pass through the screen, into the tank, the 'shorts' remain in the box on the screen bottom. The 'shorts' remaining on the screen may be washed there, using a whisk broom and a small stream of clean solution, until all of the precipitate which was deposited on them has been removed and fallen into the tank below, leaving the 'shorts' clean and ready to be placed in trays for subsequent use in precipitation.

Another method of washing the 'shorts' is to place them in a barrel cleaner, consisting of a barrel whose perimeter is made of a sheet of No. 20 iron screen cloth; whose head is made of thin sheet iron; and having an axle and handle for turning. This barrel, placed with its axle in a horizontal position, is partly submerged in a small tank, and upon being revolved, the 'shorts' rub

against each other and are entirely freed from attached precipitate, which passes through the screen and falls to the bottom of the tank, leaving the 'shorts' clean and ready for subsequent use. The 'shorts', after being cleaned, are placed in layers about 2 in. deep in trays whose bottoms are composed of No. 10 iron screen cloth. These trays are about 3 in. deep, so as to allow 1 in. of solution between trays, and are placed one above another in one of the compartments of the zinc-boxes, being made of the exact size to fit inside the compartments. In some mills the first compartment of each zinc-box is filled with trays of shorts, while in other mills all of the compartments of one box are filled with trays, and to this box is added all of the cyanide required to strengthen the mill solution; as by using a strong solution in the box containing the shorts these are more quickly dissolved.

Filtration of Precipitate Slime.—The precipitated slime, which has been collected by the clean-up into a small tank, should be filtered to separate it from the accompanying solution. In large mills it is filtered by being pumped directly from the collection tank through a filter-press, arranged with squares of coarse filter-paper over the filter-cloths, in order to facilitate cleaning, and to prevent any fine precipitate from passing through the pores of the cloth. In some mills the precipitate is washed in the filter-press, by pumping water through the cakes, but when the capacity of the press is small this washing is dispensed with, as it may be preferable to lose the small amount of cyanide contained in the solution remaining in the cakes, in order to gain capacity. The cakes are generally dried as much as possible in the press by pumping air through them, so that the cakes delivered from the press rarely contain more than 25 to 35% of moisture. In small mills, and also in some large mills, the filtering of the precipitated slime is done in filter-boxes.

Filter-Boxes.—The following description of the filter-box used in the mill of the Standard Consolidated Mining Co. of Bodie, is taken from the book, 'Practical Notes on the Cyanide Process,' written by Francis L. Bosqui:

The apparatus (Fig. 34) consists of a carefully made water-tight box, 4 ft. high, 3 ft. long, and 2 ft. wide, constructed of 2½-in. pine planks. The interior of the box is divided into two compartments by a filter-bottom, which lies 12 in. from the top. This filter-bottom, which is 1¾ in. thick, and perforated to within 1½ in. of its edge with ¾-in. holes, 1½ in. apart, is supported by a centre post and four corner posts; and any possible collapse of the sides of the box is forestalled by braces running at right angles to the centre post. To resist the strain imposed by the vacuum pump, the whole box is further strengthened by a system of ½-in. bolts, running through the timbers from top to bottom, and across the ends.

Details of false bottom.—The perforated area of the false or filter-bottom is covered by a piece of heavy wire cloth, about 15 mesh to the inch; the object being to prevent the superimposed blanket-filter from sagging down into the perforations, and to increase the filtering area by preventing close impact of the blanket with the surfaces between the holes. On top of the screen is placed one or more thicknesses of heavy mill blanketing, as the case requires. The lower blanket is pressed into place by means of a wooden frame with sides 1½ in. wide. The blanket is cut large enough to admit of its being pulled up around the frame. Into the top of the frame, in the middle of each side, is sunk a strip of iron ½ in. thick and 12 in. long. In the centre of each strip is a square hole, to receive the squared end of an upright rod. The upper end of this rod is threaded, and passes through a strip of iron

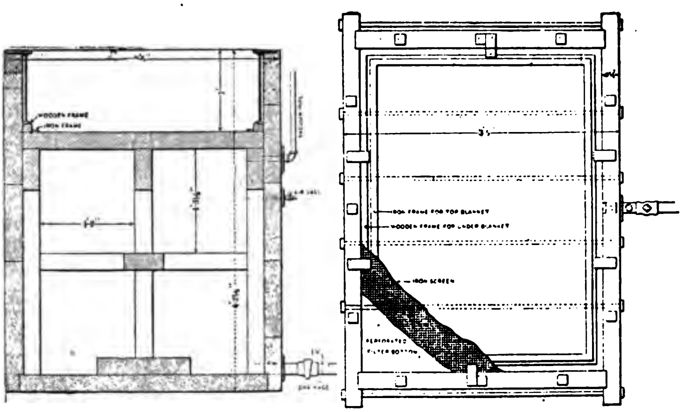


Fig. 34. Filter Box.

which projects over the inner edge of the top of the box. The rod is provided with a nut, which, when tightened up against this top strip of iron, compresses the wooden frame tightly over the blanket. The frame, with one of these rods on each side, prevents fine slime from passing down into the compartment below the filter-bottom.

Filter-blankets.—Ordinarily one blanket will suffice; but it will be found more convenient to use a second, which may be easily removed and washed, while the first may remain stationary through several clean-ups. The second blanket may be stretched in place by means of an iron frame, with sides 1 in. wide, which should be made to fit accurately within the wooden frame which holds the under blanket.

At the bottom of one side of the box is a 2-in. globe valve, for the discharge of clear fluids accumulating under the filter; and just beneath the filter is the entrance of the ½-in. exhaust pipe

coming from the vacuum pump. If steam is used for vacuum purposes, a 1½-in. steam injector may be connected. Near the entrance of the vacuum pipe is an escape-valve, which may be opened to release the pressure, when, for any reason, such as changing the blankets, etc., filtering is temporarily interrupted. For convenience, a gauge may be connected to the side of the box, to indicate the height of the liquid under the filter. This liquid should be drawn off before the surface reaches the height of the vacuum pipe. In the mill of the Standard Consolidated Mining Co. the filter-box is filled with precipitated slime by dipping the slime from the deposit tank in buckets.

In Vol. II of the Proceedings of the Chemical and Metallurgical Society of South Africa, of June, 1897, E. H. Johnson has described a simple filter for precipitated slime, which he made out of a tank 6 ft. in diameter. This apparatus (Fig. 35) has a lattice in the

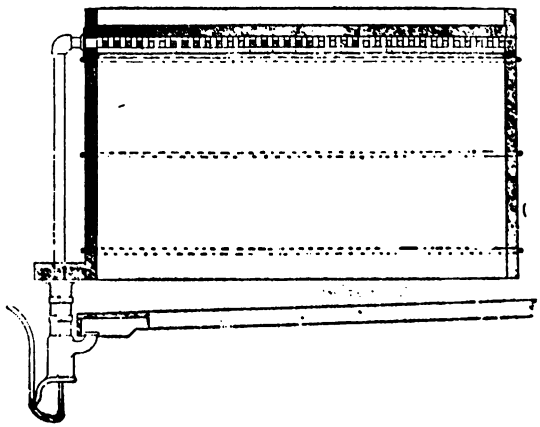


Fig. 35. Filter Tank.

bottom with one-inch square holes, covered with fine canvas cloth, as a filter-bottom.

On one side of the tank, below the filter-bottom, there is a connection for a 1½-in. pipe, through which, by means of an ordinary hand suction-pump, the solution is drawn from below the filter-bottom, and thrown into the first compartment of the zinc-boxes; where any unfiltered slimes, which it may contain, will be deposited.

Johnson states that this apparatus works rather slowly, but that the solution filters perfectly clear, and he did not care to run the pump faster for fear of breaking the filter-cloth. If he had placed some wire screen cloth below his filter-cloth it would have sustained it so that there would be little fear of breaking; and in fact, a similar filter-tank is in use in several Mexican plants, using an iron tank about 18 in. high, instead of a wooden tank, and having screen-

cloth below the filter-cloth, and the pump used is a small vacuum-pump run by machinery at a high velocity. In Mexican practice with these tanks, the solution and slime from the clean-up, after passing through a screen-box to remove the shorts, fall directly into the filter-tank, where the filtering proceeds without any difficulty.

Refining and Melting Precipitate.

After obtaining the precipitates, whether in the form of cakes or as a thick mud from the filters, there are five different methods which may be employed to convert their gold and silver contents into bullion. One of these methods is in use in every cyanide mill using zinc precipitation, and although Mexican practice confines itself almost entirely to the first method, as a matter of general information it may be well to review them all, and show their merits and defects. They are:

1. Drying and direct melting in crucibles.
2. Drying, roasting, and melting in crucibles.
3. Treating with sulphuric acid, washing, drying, and melting in crucibles.
4. Treating with sodium bisulphate, washing, drying, and melting in crucibles.
5. Smelting with litharge, and cupelling resulting lead bullion.

The details of each method are as follows:

(1) Drying and Direct Melting in Crucibles.

This treatment is very simple, and is that ordinarily in use in most of the mills in Mexico. The cake from the filter-press, or the mud from the filter-tank, may be dried by steam, introduced into the false bottom of the car containing the precipitate, or by placing in pans and heating in muffles at a low heat, or by heating over a slow wood fire, in large open sheet-iron pans placed on brick pillars. After drying the slime it is partially pulverized, by breaking up the largest lumps; mixed with the proper flux and melted in graphite crucibles. In the mill of the El Oro Mining & Railway Co., of El Oro, Mexico, the cakes from the filter-press are placed in special steam jacketed cars, where they are steam dried for from 12 to 18 hours. They are then mixed with the following flux:

	Per cent
Precipitate	100
Borax-glass	18
Calcined soda	6

This mixture is briquetted in a special press under a pressure of 2000 lbs., to prevent loss in dusting, and the briquettes are then melted in graphite crucibles. (See 'melting practice,' Chapters III and IV).

In the San Francisco mill of Pachuca, Hidalgo, Mexico, the cakes from the filter-press are dried in a round iron pan, about 5 ft. in diameter, placed on brick pillars and heated by a slow wood fire, until they contain from 5 to 10% of moisture. Then, after breaking up the larger lumps, the precipitate is mixed with the following flux:

	Per cent
Precipitate	100
Sodium carbonate	9
Borax-glass	12
Quartz tailing	2

A graphite crucible, No. 300, is placed in the coke furnace, and, as soon as it begins to heat up, is filled with the mixture of precipitate and flux. The crucible is then covered and the heat is raised until the whole of the contents of the crucible have melted. As soon as this occurs, the cover is removed and the slag stirred with an iron rod to see that no unmelted portion remains and that the slag is liquid; the liquid slag is then removed with an iron spoon until very little slag remains on top of the molten metal; a small quantity of the cooled slag, which was first removed from the crucible, is then thrown back into the crucible to chill its contents, and on top of this cold slag a further charge of precipitate and flux is placed. filling up the crucible. This is again allowed to melt, and the operation of removing the slag and filling up with a new charge is repeated until the crucible is half full of molten metal, or until the whole of the charge has been melted. After the last charge has been added, and melted, the liquid slag is removed as much as possible with the spoon. Then a handful of bone-ash is thrown into the crucible to thicken the remaining slag which may then be removed by means of an iron spiral. Then add a handful of borax glass, and after this has melted, another handful of bone-ash so as to thicken and remove the borax, with the spiral, as before. This operation is repeated several times; the refined bullion is then poured into moulds.

Borax-glass has the property of dissolving zinc oxide, as soon as it forms on the surface of the molten bullion and by repeating the operation just described, it is possible to obtain very clean bars. The precipitate of the San Francisco mill contains from 40 to 60% of gold and silver, while the bars obtained by the treatment described assay from 920 to 980 fine gold and silver. All of the slag obtained by this treatment is saved for subsequent treatment.

Julian and Smart propose the following flux, stating that it is in use in various mills, and that by its use it is possible to obtain bars from 600 to 750 fine:

	Parts
Dry precipitate	100
Sodium bicarbonate	50
Borax	35
Sand	15
Nitre	2 to 4

The above proportions may be admissible in melting gold precipitate, but where silver precipitate is handled, whose weight in some mills is from two to four tons per week, such a large proportion would be too expensive.

The disadvantage of the system of direct melting consists in the fact that the zinc present in the precipitate will volatilize at high temperatures, and may carry with it some of the gold and silver. Furthermore, in mills where the practice of briquetting the precipitate and flux is not followed, the loss in dust escaping up the flue while charging the crucible may be considerable. These losses in large mills, where proper attention is given to the matter, where briquetting is practiced, and where oil-furnaces are used, in which the flame is shut off and the dampers closed while charging, are reduced to a minimum; but in plants where these precautions are not taken, although the losses cannot be estimated with exactness, owing to the impossibility of obtaining an exact sample of the precipitate, it is believed that these losses may at times amount to from 1 to 5 per cent.

(2) Drying, Roasting, and Melting in Crucibles.

In this treatment the damp precipitated slime, after filtering, is mixed with a small quantity of niter, and placed in roasting-dishes in a muffle-furnace, having a free entrance for air to oxidize the zinc. When the mass is dry, the heat is raised to such a point that the greater part of the zinc is oxidized, the fumes of zinc oxide being drawn off by a chimney with a well regulated draught.

The oxidized precipitate may then be melted with the following flux, obtaining bars 800 fine, according to Julian and Smart.

	Parts
Oxidized precipitate	100
Sodium bicarbonate	40
Borax	40
Silica	15

The following modification of this treatment was recommended by W. A. Caldecott in the Jour. Soc. Chem. Ind. Volume XVII of 1897: Dry the slime to a dry powder; then mix them with silica and nitre in the following proportions:

	Parts
Dry slime	100
Quartz sands	30
Nitre	30

Then spread out the mixture in sheet-iron trays to the thickness of ¾ in. and heat just above the boiling point of water. Then touch a piece of burning charcoal, covered with a little powdered nitre to the mixture; the whole mass will enter into combustion, which proceeds without decrepitation, but produces an intense heat, and causes the evolution of dense fumes of zinc oxide, leaving a

half fused mass which is not so likely to give rise to loss in dusting during the subsequent melting. It is supposed that during this operation the base metals, as well as any organic matter contained in the slime, are oxidized, and some of the sand combines with them, forming fusible silicates. The product of this operation should be melted with the following flux:

	Parts
Oxidized slime	100
Borax-glass	40
Dry sodium carbonate	10

This flux is said to give a clean liquid slag, while the bars produced are 800 fine. Both of these methods for roasting slime are liable to considerable losses of gold and silver, carried off with the zinc fumes, and as they require double manipulation, these treatments are rarely employed at present, being substituted either by the preceding, or by one of the following methods:

(3) Treatment with Sulphuric Acid, Washing, Drying, and Melting in Crucibles.

This treatment was originated in order to dissolve the short-zinc which existed in precipitated slime, so as to overcome the loss of gold and silver which results from the roasting or direct fusing of slime containing a large quantity of zinc; loss in dusting due to the various manipulations, as well as loss in volatilization of the gold and silver with the zinc oxide. The treatment consists in mixing the damp slime with sulphuric acid, in wooden tanks, followed by a thorough washing, to separate the sulphate of zinc thus formed, filtering, drying, and melting.

South African Practice.—E. H. Johnson, in Vol. II of the Proceedings of the Chemical & Metallurgical Society of South Africa, proposes the apparatus represented in Fig. 36 for this purpose.

It consists of a wooden tank, 6 ft. in diameter, arranged with a cover and a wooden stirrer which may be removed from the apparatus at the conclusion of the chemical reaction. A 3-in. pipe attached to one side of the upper portion of the tank carries the fume outside of the house where the treatment is given. The operation, according to Johnson, is as follows: After filtering the slime, in the apparatus shown in Fig. 35, "a water-wash is then added and pumped through until the slime is free from cyanide. The gross weight of the slime, including moisture, is then taken, by weighing the buckets of moist slime during transference to a large sheet iron tray placed alongside the acid-tank. The object of weighing is to determine the amount of sulphuric acid necessary to destroy the zinc. Having found the approximate weight of slime to be treated, sufficient water is run into the acid-vat to form, on the addition of the acid, a 10% solution. He used one pound of

acid for every pound of moist slime, with good results. This would be equivalent to about one and a third pounds of acid to the pound of dry slime. Having the requisite amount of water in the vat (Fig. 36) the weighed quantity of acid is then added and the vat closed down tightly. The stirring apparatus is kept continually moving during the time of feeding in the slime, which is fed in gradually in the same condition in which it was taken from the filter-vat. He found it beneficial to keep up a continual stirring for at least half an hour after the action had apparently ceased.

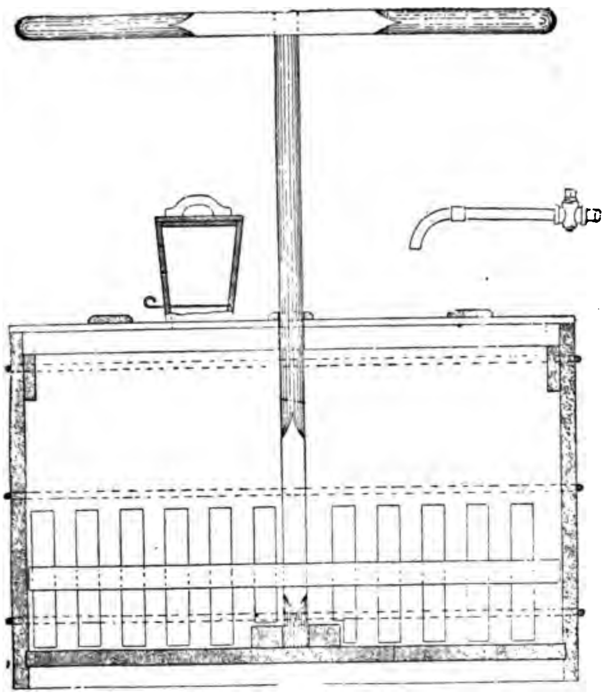


Fig. 36. Tank for Treating Precipitates with Sulphuric Acid.

"The feed-hopper has a slide door of sheet-iron in addition to the closely-fitting cover on the top, so that by keeping the slide closed a quantity of slime can be fed into the hopper and the cover closed before letting the slime fall into the acid. This is to keep the fumes from the operator as far as possible, as the fumes are distinctly irritating. After all the slimes are in the acid, a jet of water is turned into the hopper to wash down any adherent slime. The stirring apparatus is then removed and well washed in the vat during removal. The vat is then filled with water and allowed to settle. Working with dilute acid and not heating he obtained a perfect settlement within an hour. When heating with

a steam jet settlement was much more difficult. The washing is done by syphoning off the clear liquor, and filling the vat repeatedly with water, until the solution is neutral to litmus paper—usually four or five washings. It is well stirred at each refilling by means of a long wooden paddle, a rotary motion being given to the water. This causes the slime to collect in the middle of the vat and reduces the risk of loss during syphoning, the syphon being let down at the side. A sample of the washings taken continually during syphoning off showed, on careful assay of a large sample, to contain 13 grs. of gold per ton of solution. The drying of the resultant gold slime is conducted on an open drying-hearth in large cast-iron enameled dishes. The cakes are subsequently broken up and transferred to small sheet-iron trays in thin layers, and subjected to an increasing heat. When cool the slime is ground, fluxed and transferred to the crucible. It fuses quietly and with but little fume, and normally yields 50 to 60% of the weight of slime as bullion."

The average fineness of the bullion obtained was 820 fine. The flux used in melting consisted of:

	Parts
Dry slime	100
Borax-glass	66
Sodium carbonate	9
Fluorspar	9

The cost of the reduction to bars, in acid, flux, coke and crucibles was 6.7d. per fine ounce, which is equivalent to $4.31 per kilogram.

American Practice.—The following description of the practice in the mill of the Standard Co. in Bodie, is taken from Francis L. Bosqui's book 'Practical Notes on the Cyanide Process.'

In this mill the cyanide solution is replaced in the zinc-boxes by pure water before the clean-up. The compartments are washed one by one, until all of the precipitated slime and 'shorts' have passed into the acid-treatment tank. The contents of each compartment are allowed to settle in the acid-tank, for an hour, while cleaning the shavings in the next compartment, and then, before introducing the contents of the second compartment, the supernatant liquid is siphoned off into a settling-tank, by means of a 1½-in. rubber hose; repeating the operation with the contents of each compartment as they enter into the acid-tank. When all of the compartments have been cleaned, and the supernatant liquid removed as much as possible, the acid treatment is practiced as follows: "A sliding door in the roof directly above the tank is opened, as well as a door near the opposite end of the clean-up room, to insure a good draft to carry away the fume. A conical Russia-iron hood, arranged so that it could be raised or lowered, was tried and discarded; it did not appear to furnish a wide enough channel to carry away the vast amount of vapor evolved by the first contact of the acid with zinc. When everything is ready, a bucketful of acid

(weighing about 30 lb.) is thrown into the mass of wet slime. The first effervescence is usually very violent, especially if the cyanogen residues have not been properly displaced from the boxes before cleaning up; but if the draft is good, the fumes rise in a straight column, and the operator may stand alongside the tank without inconvenience. The slime is from time to time stirred up with a long wooden hoe until the action of the first charge of acid somewhat subsides. From now on, only half a bucketful need be used at a time; and with the addition of every fresh charge about the same quantity, in bulk, of hot water may be run in from the faucet over the tank. This process is continued, about 15 lb. at a time, with occasional stirring, until further addition of acid produces only a mild effervescence. The mixture may then be left undisturbed for about two hours. A quiet effervescence will continue until all the small particles of zinc are consumed. The end of the reaction may be determined by adding a little acid. If no effervescence takes place it may be taken for granted that virtually all the zinc has been consumed. At Bodie, where from 1,500 to 3,000 oz. of precipitate are refined at each clean-up, the actual consumption of zinc by acid requires from four to six hours. The time required will obviously depend upon the quantity of zinc present, the shape of the acid tank, the quality of the acid, etc. When the effervescence has subsided the black, thick mixture in the tank is diluted to within a few inches of the top, with hot water—a slow stirring being kept up while the tank is filling. The object of this second step of the operation is to wash out the zinc sulphate residue which occurs as a product of the acid reduction of the zinc. If allowed to remain it very materially increases the difficulties and cost of melting, and may cause some loss of gold from volatilization. Consequently, upon the thoroughness with which the washing is done depends much of the success of this mode of treatment. The tank thus filled is allowed to stand from four to six hours, until the supernatant liquor may be siphoned off almost clear into the settling tank. A second wash of water is then applied, and a series of such dilutions kept up, until the sulphate of zinc is practically eliminated. From five to eight dilutions will usually be necessary. When the final dilution is drawn off down to within a few inches of the precipitate, the residue may be discharged into the filter-box (Fig. 34). The wooden hoe may be used to push the material along the inclined bottom of the tank to the discharge valve. Any adhering material may be carried down by a jet from a small hose, and the tank thoroughly cleaned out."

The precipitate, after filtering, is dried in an iron pan over a brick furnace, cutting the contents of the pan into small sections with a spatula, to allow the escape of steam, and the material is not turned over or disturbed while drying, thus avoiding loss in dust. It requires from 12 to 15 hours to dry from 150 to 200 lb. of pre-

cipitate over a slow fire, and the cost of the acid treatment in the Standard mill, per ounce of gold recovered is as follows:

	Cents
Sulphuric acid	4.74
Wood consumed for steam vacuum ejector and for drying precipitate	4.18
Labor	2.71
Total	11.63

or $3.73 per kilogram of gold recovered.

In melting the precipitate in this mill, the mix 100 oz. of precipitate with from 50 to 75 oz. of flux composed of:

	Parts
Borax	4
Soda	2
Quartz sand	1

The cost of melting the precipitate, including charcoal, flux, crucibles and labor, was 15.58 cents per ounce of gold recovered, or $5 per kilogram. The treatment of precipitate with sulphuric acid is not only expensive, but is dangerous, and on various occasions the generation of hydrocyanic acid and arseniuretted hydrogen, which generally occurs when treating ores containing arsenic, or when using sulphuric acid containing arsenic (a very frequent impurity of commercial acid.) have resulted fatally. On one occasion four high-class employees, who were present during an acid clean-up, died from the effects of the gases inhaled.

(4) Treating with Sodium Bisulphate, Drying, and Melting in Crucibles.

G. W. Williams, in the Monthly Journal of the W. A. Chamber of Mines of June 29, 1907, states that this treatment is used in South Africa, and that it has the advantage of being cheaper and less dangerous than the sulphuric acid treatment, as sodium bisulphate is a secondary product of the dynamite factories which they are pleased to sell at any price, and as this salt is absolutely free from arsenic the danger from the presence of the very poisonous arseniuretted hydrogen gas is avoided. The solution of the zinc by the excess of acid contained in the bisulphate requires the use of larger tanks than those required in the sulphuric acid treatment, as the bisulphate only acts well when dilute. The tanks used for the purpose in South Africa are lead-lined, with a wooden cover, through the center of which passes the shaft of an agitator moved by pinions, and have two chimneys for the escape of the gases formed during the treatment.

The treatment is similar to that already described under acid-treatment, except that, after the solution of the zinc, they add a

solution of 2 lb. of glue in a gallon of water, in order to assist in the settlement of the precipitate, in case it does not settle well otherwise.

The following analyses represent the composition of the precipitated slime, which have been treated with sulphuric acid or sodium bisulphate and filtered, before and after roasting:

ANALYSES OF PRECIPITATED SLIME.

	Before Roasting Per cent	After Roasting Per cent
Gold	35.8	37.0
Silver	3.7	3.8
Metallic zinc	2.4	....
Zinc oxide	6.0	7.5
Metallic lead	19.1	....
Lead oxide	....	20.0
Metallic copper	2.2	....
Copper oxide	.....	2.2
Iron oxide	1.0	2.5
Metallic nickel	0.5	....
Nickel oxide	....	0.5
Ferrocyanide	2.9	....
Lime and magnesia	trace	trace
Sulphuric acid	6.0	8.0
Sand	12.5	14.0
Total	92.1	95.5

The lead content of the slime was due to the use of the lead-zinc couple in precipitation. The ferrocyanides were converted into iron oxide after roasting. A part of the sulphur which appears as sulphuric acid was in the form of lead sulphide. The sand present in the precipitate is due to imperfect filtration before precipitation. When copper, nickel, and mercury are present in the solution, they are usually precipitated on the zinc, and as the sulphuric acid does not attack them under existing conditions, they will be found in the precipitate.

Mr. Williams recommends the following flux for melting the roasted precipitate:

	Parts
Roasted slime	100
Borax-glass	30
Soda	5
Quartz sand	20
Manganese binoxide (commercial)	20
Nitre	7
Fluorspar	5

The object of the oxidizing agents, manganese binoxide and nitre, being to oxidize the metallic zinc so that it will dissolve in the borax and fluor spar. In some places they substitute one-half of the manganese binoxide by its equivalent in iron oxide in order to obtain a more fluid slag. When melting with the above men-

tioned oxidizing agents it is necessary to use clay-lined graphite crucibles, as otherwise the oxidizing agents will attack the graphite of the crucibles and destroy them. An excess of manganese binoxide should not be used, as it not only forms a very thick slag, but also may dissolve some of the gold and silver, thus increasing the loss in melting. For ordinary work nitre is the best oxidizing agent, and in some places, where the bars resulting from the first melt are too low-grade to ship, they are refined by remelting in a clay-lined graphite crucible and allowing a small stream of powdered nitre to fall on the centre of the molten metal. Then adding borax-glass to form a slag with the metallic oxides thus formed. This slag is removed, and the operation repeated until the slag formed by the addition of borax-glass has a pale green color, when it will be found that the bullion is quite fine.

In other places the base metal contents of the bullion are refined by introducing a stream of compressed air into the molten metal, through a porcelain tube, which passes below the covering of slag. This slag, composed of sand and borax-glass, dissolves the metallic oxides as they are formed and prevents any loss of the boiling metal. Zinc oxide is very soluble in borax-glass, which in the presence of quartz sand forms the boro-silicate of zinc. The use of boric acid has been proposed, and it is probable that it would have a greater efficiency as a solvent for zinc. Its use in practice, however, will depend on the price. Fluorspar, when melted with quartz-sand and zinc, forms the fluo-silicate of zinc, which is even more liquid than the boro-silicate, and, as its cost is but the tenth part of that of borax-glass, it makes a very efficient and economical flux, although, unless an excess of silica is present, it will attack the graphite crucibles.

Advantages and Disadvantages of the Acid-treatment.—The advantages claimed for this treatment, whether using sulphuric acid or sodium bisulphate, are that the resulting bars are high-grade; that the losses in dusting are less, as the operations are performed in solutions; and as the zinc is removed from the precipitate before melting, and by solution rather than by roasting, the losses of silver and gold by volatilization (whether such losses occur during the melting, as is possible in the direct-melting process, or during the roasting, when practicing roasting before melting) are thereby avoided.

The disadvantages presented in the acid treatment are the danger of poisoning; the cost of the treatment, not only in the consumption of acid, and extra labor, but also in the consumption of zinc 'shorts' which still have a value as precipitants; and the probability of losses of gold and silver during the treatment. These losses, according to various authors, occur in three forms: first, in metals dissolved in the acid; second, in gold and silver lost by volatilization, both during the dissolution of the zinc, when, it is stated, the hydrogen gas, or hydrocyanic acid, which are evolved, carry

away some of the gold in a volatile form, and also during the roasting of the acid-treated slime, which is sometimes practiced; and third, in gold and silver dissolved in the matte which is formed when melting acid-treated slime which have not been roasted. This matte is always formed when melting acid-treated slime formed by precipitation on the lead-zinc couple, or when the slime still contains some zinc sulphate which has not been removed by washing after the acid treatment, as the lead sulphate formed by the action of the sulphuric acid on the lead, or the zinc sulphate which may be present, when melted with a reducing agent, such as the cyanides and ferrocyanides which may be present in the precipitate, are converted into the sulphides of lead and zinc, which combine with the copper, iron, silver and other metals present in the slime, forming the complex sulphide called 'matte.' This matte is often very rich in gold and silver, and in order to prevent its formation, it is customary in many mills to roast the slime after acid treatment, in order to oxidize and volatilize the sulphides before melting the slime.

In order to determine the loss which occurs during this roasting, tests were made by various members of the Chemical and Metallurgical Society of South Africa, whose results were published in Vol. III of the Proceedings of that society. In one lot of 2000 oz. of precipitate, thoroughly mixed after acid-treatment, one-half was melted directly, and the other half was roasted by heating to a red heat before melting. The result of the two operations gave 3% more gold from the direct melt, than from the melt after roasting, and as these tests were verified by several members of the Society with similar results, it is considered as proven beyond a doubt, not only that this loss occurs, but that it is a source of a loss of thousands of dollars yearly to the companies employing this method. When it is considered that the 3% loss above mentioned is simply the result of one portion of the acid-treatment, and it is known, as previously detailed in this paragraph, that there are other sources of loss in acid-treatment, it may readily be credited that the enormous loss of 10% of the total value of the precipitates, which was observed in one instance by P. S. Tavener (see next paragraph), has occurred in many mills using this treatment.

(5) Smelting with Litharge and Cupelling the Resulting Lead Bullion.

This treatment was introduced by P. S. Tavener and described by him in a paper entitled 'The Lead Smelting of Zinc-Gold Slimes' which appeared in Vol. III of the Proceedings of the Chemical and Metallurgical Society of South Africa, from which the following drawings and description are taken.

It consists of two distinct operations: first, smelting the precipitate slimes, mixed with litharge, flux and some reducing agent, in a reverberatory furnace, to collect the gold and silver in lead

bars; second, the cupellation of the lead bars in a cupelling furnace, by which operation the gold and silver are obtained, free from impurities, while the lead is again converted into litharge, which may be used with the next charge in the reverberatory furnace.

The smelting furnace used for this operation is that shown in Figs. 37 and 38, and the procedure is described in the following paragraphs, essentially in Mr. Tavener's own words.

The 'clean-up' is conducted in the ordinary way with the exception that all the precipitate is at once pumped from the 'clean-up' tub into the filter-press; the fine zinc which remains at the bot-

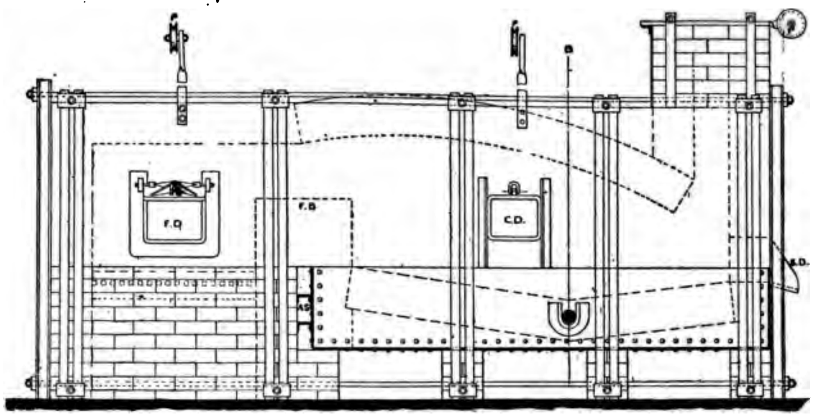


Fig. 37. Longitudinal Section of Smelting Furnace.

F D, fire door; C D, charge door; S D, slag door; A S, air space; F B, fire bridge; P, pulley for counter weight 1.

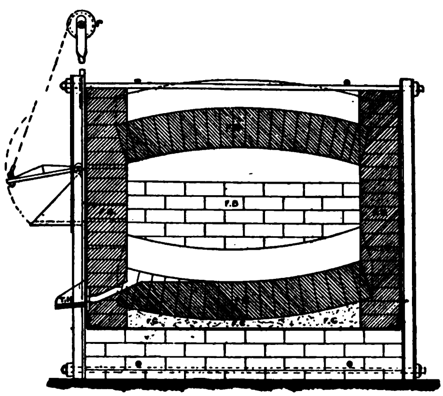


Fig. 38. Cross-section of Smelting Furnace.

T H, top hole; F B fire brick; F G, fire brick grouting.

tom of the 'clean-up' tub is heaped up on one side and allowed to drain for about half an hour, and is then ready for the smelting room. The filter-press is cleaned out and the cakes taken in their moist condition to the furnace, and there both slime and fine zinc are put in trays into a drying oven, and sufficient time allowed to warm through; fifteen minutes for each tray being sufficient. Care is taken to keep the fine zinc separate from the filter press slime, and on no account should they be allowed to get mixed.

In charging the furnace the slime is first dealt with. After warming in the drying oven, it is at once rubbed through a sieve—four holes to the linear inch— and then roughly weighed for fluxing, the necessary flux having been previously mixed. The slime is fluxed and passed through a screen to insure thorough mixing. It is then shoveled direct into the furnace. When all the filter-press slime is fluxed and charged, the fine zinc is dealt with in the same way and put into the furnace on top of the slime in order to prevent loss by dusting, and also to have the greater portion of litharge present on the top of the charge. The fluxes used are: residue assay slag and commercial litharge (the former costs nothing, for unless used in this manner it would be thrown away).

I have found that the following, with a little variation, will give satisfactory fusion and clean slag: Slag 25 to 30% made up of 10% assay slag, the balance, equal quantities of old slag and scalings from the pots of previous melts. The quantity of litharge used varies, and is influenced by the value of the slime, the weight of charge in the furnace, and the percentage of gold required in the resulting lead bullion. In fluxing, the amount of litharge used is from 100 to 150% of the weight of fine zinc in its partly dried condition. With slime 60% is sufficient, and this can be reduced to 40%, if there is sufficient charge in the furnace to give 12,000 oz. lead bullion. Roughly the proportion of fluxes are as below:

	Parts
Slime	100
Litharge (PbO)	60
Assay slag	10 to 15
Slag previously used	10 to 15
Silica or quartz (SiO_2)	10 to 20
Sawdust	1 to 2

	Parts
Fine zinc	100
Litharge	150
Slag	20

The product to be melted will vary at different mines, and no hard and fast rule can be laid down for the different fluxes. A few trials made on a small scale in an assay office would quickly determine the correct proportions to give satisfactory results. In practice it is found that considerably less assay or other slag will be required than that necessary to obtain a good fusion by melting a

small quantity of slime or fine zinc in a crucible; and it would be safe to use 30% less slag when melting in the reverberatory furnace. I endeavor not to allow the lead bullion to carry more than 8% of fine gold as a maximum, it being preferable to make a larger quantity of lead than to have it very rich, the limit being say 10%. With the slime the only other addition to the flux is a reducing agent, sawdust for instance. One per cent of sawdust on the weight of litharge is added, or if more than 60% of litharge has been used, 1½ to 2% sawdust is necessary. To the fine zinc none is added, since it is better to rely on the zinc to reduce the litharge, and also advisable to insure an excess of litharge in the slag.

When the entire charge of zinc and slime is in the furnace, it is banked up from the sides to the centre so as to avoid the possibility of particles remaining on the sides above the slag level as the charge reduces and settles down. A covering of litharge is spread over the surface, and on this again a light covering of easily fusible slag is spread. The furnace is charged the day previous to melting, and one of the night-shift men lights a slow fire about 3 a. m.—this serves to dry the charge. At 5 a. m. the damper is opened and the fire urged and in half an hour the furnace is at a melting heat. By 9 or 10 a. m. the charge is reduced, then sweepings from cyanide-works, melting-room, or any slag requiring re-smelting is added, and is quickly absorbed in the molten bath. When all this has been fed in and melted, and the slag become fluid, it is well stirred with a rabble, and sawdust is thrown in to reduce the excess of litharge in the slag. The slag is now run off into pots through the slag-door. It will be seen from the plan that the level of this door is 4 in. above the centre of the lead bath—a bath of 12,000 oz. of lead bullion almost occupies this space.

Before filling the furnace the slag-door is built up about 12 in. by placing the flat cast-iron plates ½ in. thick, bedded in fire clay one on top of another, and in front of these plates a bank of fire clay is also made. In order to run the slag off, all that is necessary is to break away this bank, plate by plate, and so allow the slag to flow over into the pot. When the pot is full it is wheeled away and another is put in its place. The filled pot is run outside, and after standing a minute or two, is tapped and the molten slag allowed to run out on the ground to cool; that which remains on the sides and bottom of the pot is brought back for further use. When no more slag will flow from the furnace owing to the bath being down to the level of the slag door, it is removed by rabbling. At first sight it would appear difficult to draw this remaining slag off without dragging out some lead, but a very little practice enables it to be done so closely that there is little but a thin skimming of slag remaining. In the event of a little lead being pulled out, it is recovered from the slag pots. It is for this reason that the pots are tapped about 2 in. from the bottom. By opening the

fire-door this last skim on the lead bath quickly thickens. A shovelful of lime is thrown in to assist. The skim is easily pulled off, and of course is held over until the next melt. By this means a clean surface of lead is exposed, and any zinc present would be quickly got rid of, for at this stage the lead is at a bright red heat, and the free access of air due to the open fire door quickly oxidizes it. So far, the lead recovered by this method has always been clean and soft, a proof that no zinc could be present since 1% zinc gives to lead a distinct silvery color and makes it so hard that it cannot be rolled. The lead bullion is tapped by driving a ½-in. steel bar tapered to a point into the tap hole, which is closed with a fire clay plug, and the lead is run into an iron trough which conveys it to the moulds, these having been previously placed together on the floor. Before tapping the furnace the lead bath is well stirred, and with a ladle a sample is taken and granulated, by pouring into a bucket of clean water. This sample is absolutely reliable, and reduction works will pay on 100% of assay value on such a sample.

The mode of conducting the process of cupellation is next to be dealt with. The first operation consists of making the test, which is an oval cast-iron frame filled with bone-ash ground sufficiently fine to pass through a 20-mesh sieve. Other materials, such as a mixture of limestone and fire clay, three parts by volume of limestone to one and a half fire clay, ground through a 12-mesh screen, would serve the purpose, but I prefer bone-ash. The bone-ash before being put into the test-frame is prepared by dissolving in water 1 lb. of caustic potash to 33 lb. of dry bone-ash. The test as shown in the sketch (Fig. 39) and used on the Bonanza, requires 500 lb. to fill it. The bone-ash in the test after cavity has been cut out weighs 300 lb., and is moistened to just such a point that by squeezing into a ball in the hand it will break clean. A good deal depends on correct moistening, 10 to 11% of water on weight of bone-ash has been found by practice to give the most satisfactory results. To the inexperienced, the danger is always to make it too plastic, and it is better to have it too dry than the least bit too moist. When it has been damped, it is passed through sieves, 10 holes to linear inch, to break up all lumps. The test-frame is now placed on a 2-in. cast-iron plate bedded in cement, bone-ash is shoveled in until the frame is quite full, and is then tamped in. When filled first it is best to tread it down, starting from the center and working round evenly. It is then leveled off, and lightly and evenly beaten with tampering irons, after which two shovelfuls of bone-ash are beaten in at a time until the whole test is full. The surface is then hollowed out with a tool (made by cutting a mason's trowel in half), leaving a rim 3 in. wide at the back and sides, and at the front 10 to 12 in. wide, to allow space for the formation of litharge channels. The cavity thus formed is from 2½ to 3 in. deep. It is advisable to leave the cavity slightly lower in the

front than at the back. Four 1½-in. holes are bored through at a distance of 2 in. from the test ring, and 4 in. apart, in the front of the cupel, which may be done with a half-moon shaped bit used with a carpenter's brace. The operation is now complete, and the test is put on one side in the melting-room for at least two weeks—the longer the better. It is advisable to place it near the melting-furnace so that the drying is assisted by the heat of the furnace during the melting. Such a test properly made is capable, when cupelling once a month, of handling four months' output, dealing each month with a ton and a half of lead. When a new test is placed in the furnace, a slow fire is kept burning for three or four

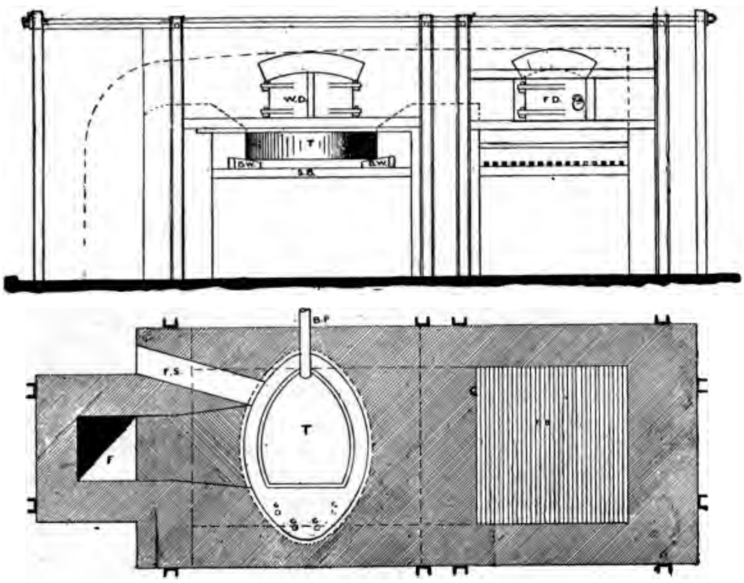


Fig. 39. Cupellation Furnace.

F D, fire door; W D, working door; T, test; B W, brick work; S B, supporting bars; F S, feed shoot; B P, blast pipe; F B, fire bars; F, flue.

hours before starting to refine. It is put in place, and held there by brick wedges resting on the two supporting bars (see sketch). An iron blast-pipe, which is of 3-in. diameter, flattened and turned down at one end to allow the blast to strike down on the molten lead, is now fitted in at the back end of the cupel (see sketch). The test having been well warmed, the temperature of the furnace is raised sufficiently to melt the lead bars which are now fed in. This is done at first by placing directly in the cupel six bars through the working doors; when these are melted the temperature is increased, and bars are then pushed in through the feed-chute, far enough to project a few inches over the lead, and as each bar

gradually melts it is pushed farther along towards the fire until entirely melted. Lead is thus fed in until the molten bath almost reaches the level of the litharge channel—this channel, ¼ to ½ in. deep by 1½ in. wide, being cut in the meantime. The temperature is now increased to the melting-point of litharge, and as soon as the lead is covered with molten litharge the blast is turned on, care being taken to regulate it by the damper, so that little ripples or waves on the surface are caused, yet not sufficient to produce spitting of the lead. The channel is previously stopped with a small plug of bone-ash, and as soon as the litharge commences to flow over it, a scraping iron is used to remove the bone-ash plug and allow the flow to continue along the channel, through the hole, and so down into the litharge pot, which is similar in shape to the slag pots, only much smaller, being only 12 in. diameter and 8 in. deep. The rate of flow of litharge is controlled by the quantity of lead melted off from the bars pushed through the feed-chute, and the feed is so regulated that about two-thirds of the lead remains covered with litharge. This operation is continued until all the lead has been melted and the gold so far refined that there only remains the concentrated bath in the test. At this stage the temperature of the furnace must be considerably increased owing to the gold-lead alloy becoming less readily fusible on account of its richness. At the same time the channel for the outflowing litharge has to be gradually deepened to allow the remainder of it to flow over, and this operation must be continued with the greatest care until, by the practically complete elimination of all lead and impurities; nothing remains but gold bullion. It is advisable, after making each cut in channel, to allow the flow of litharge to almost stop before making the channel deeper. Nearly all the copper originally in the lead bullion comes with the litharge just before completion of the operation, and the latter is thereby rendered so thick and heavy that it requires the highest temperature available, and may often require to be assisted by scraping it from the channel into the hole. If any difficulty is found, due principally to deficient temperature, this may be overcome by stopping the channel with a plug of bone-ash, and placing on top of the molten bath a ladleful of assay office slag, and stopping the blast in the meantime until the slag is quite fluid. Then by again turning on the blast and removing the bone-ash plug the litharge will again flow readily. Just previous to the point of completion there is danger of the alloy freezing directly under the cold blast, which always happens when the gold is cleaned, and is a good indication that the operation is complete. As soon as this has occurred the blast is shut off and the fire urged until the bath is once again re-melted. If the refining is complete very little litharge and slag is seen on the surface. The blast is again turned on if necessary, and this last impurity blown off. The gold now has a clean surface and will reflect a working tool held over it. Six or eight pounds of assay-slag are then added,

and after being melted this is run off. By this means a bright clean surface is given to the gold, and it should maintain this appearance after being kept molten for a few minutes. The fire door is now opened, and the molten gold is allowed to cool in the furnace to such an extent that it will not crumble when a crowbar is inserted under the cake, which is lifted up, broken in half, withdrawn through the working door, and caught in a slag-pot. No unnecessary delay must take place when the gold is set, otherwise it will be difficult to break up after removal from the cupel, and the breaking is necessary to enable it to be fed into crucibles, remelted and cast into bars.

I will now endeavor to point out what appear to be the most apparent advantages to be gained by lead smelting, as against sulphuric acid treatment. They may be tabulated as, (1) saving in cost of melting; (2) no by-products; (3) less liability of loss in handling; (4) more gold actually obtained, as shown by reliable comparative tests:

(1) The first can best be demonstrated by drawing attention to the particulars of the first Crown Deep trial. The fact of equal weights having been treated by the methods allows the making of a better comparison.

Crown Deep Costs for Acid Treating Calcining and melting		Lead-smelting method	
721 lb. sulphuric acid at 3¾d. per lb.	£11-5-6	10 bags coal at 4s. 6d. per bag	£2-5-0
Crucibles, clay liners, stores, fluxes, etc.	11-0-0		
Coal	2-18-6		
	£25-4-0		£2-5-0
443.727 oz. melted gold.		552.452 oz. gold recovered.	
31.253 oz. by-products.		Smelting cost per oz. 1d. fine.	
474.980 oz. gold recovered.		....lb. slag, assay value 1 oz. per ton (this is not credited.)	
Cost per oz. 1s. 1d. per fine oz. of melted gold recovered.			

In the above, no labor costs are included, but the labor was considerably less on the lead smelt.

During a period of four months the melting cost on the Bonanza, Ltd., exclusive of labor, but inclusive of firebrick and furnace repairs, while using the lead-smelting and cupelling process, was less than 3d. (or 6 cents) per fine ounce recovered. The gold output during this period was 12,810 oz.

(2) The value of a method that will do away with by-products is an important consideration. In this process any slag or litharge which carries precious metals will be treated in the subsequent melting, and all gold will be in the shape of fine bars, which bring

the highest market price; whereas in any other process, gold and silver will be locked up in slags, which have to be saved until the accumulation may be sold to smelters, where a treatment charge and deductions will cause these metals to be sold at about 75% of the value which might be obtained by this process, and as from 1 to 2% of the total output of a plant is locked up in these by-products it is a matter worthy of consideration.

(3) There are no acid solutions to run to waste, and as the precipitate is only handled once, and then in a moist state, there is less dusting, and a minimum of loss in handling, and most important of all, the calcining is done away with.

(4) Mr. Tavener made a large number of comparative tests, but the following will be sufficient to illustrate the results he obtained:

Equal weights		Oz. of fine gold recovered	Oz. of fine gold recovered per lb. dry slime
Part 1—After sulphuric acid and calcining weighed 2110 oz. and gave fine gold	224.272 oz.		
Add gold in slag	36.8	261.072	0.6186
Part 2—Smelted with lead and cupelled gave	282.683 oz.		
Add gold in cupel	8.35	291.033	0.6896
Part 3—After sulphuric acid and calcining, weighed 2030 oz. and gave fine gold	241.551 oz.		
Add gold in slag	36.8	278.351	0.6596
Part 4—After sulphuric acid and calcining, weighed 1875 oz. and gave fine gold	212.701 oz.		
add gold in slag	36.8	249.501	0.5912
Slag from Bonanza Smelting, say 630 lb. at 1 oz. 5 dwt. per ton			0.393 oz.
Slag from Acid Smelting, 700 lb. at 1.01 per cent			110.9 oz.

Lead process 10 per cent higher recovery than 3 acid treated lots.

"It may be argued that owing to the difficulty of thoroughly mixing the gold slime and fine zinc it is impracticable to insure that by taking equal parts, equal quantities of gold are represented in each half. The large differences obtained were attributed to this cause, but the law of averages should sometimes give the acid-treated method a high recovery. This has not happened once in the experiments made." The balance in favor of this process varied from 1 to 11 per cent.

Proposed improvements in this method.—Any possible loss in dusting would be still further reduced by briquetting the slime and flux before melting. Furthermore, such briquettes could be dried in an oven and then added directly to the molten mass in the melting-furnace, after the original charge had settled, thus increasing the capacity of the furnace. By using the 'shorts' as precipitants, placed in separate trays, as described in the first part of this chapter, the zinc-content of the charge would be a minimum, thus lessening possibility of volatilization. By using the modern cupelling furnace with a tipping hearth, the bullion after refining could

be poured direct into molds, thus avoiding the cost and possible losses of a second melt.

Application of this process to Mexican practice.—The process described by Mr. Tavener so far has not been applied in Mexican practice, and the fact that a smaller quantity of flux is required in melting silver precipitate than that used in the American and African mills for melting the gold precipitate, on account of the greater fusibility of the silver precipitate, may cause a less difference in cost than that obtained by Mr. Tavener, especially as the general practice in Mexico is that of direct fusion without acid treatment. The losses by volatilization are also probably less when treating silver precipitate, but the values tied up in slags are a defect in direct melting, although reduced to a minimum by remelting the shells from slag pots, as explained in Chapter IV. In one Mexican mill, while melting about 5 metric tons of precipitate which yielded 2555.4 kg. of fine silver and 27,430 gm. of fine gold, the cost in crucibles, coke, and flux (excluding cost of labor) was \$690, or less than ½ cent per fine ounce of bullion produced, and it is questionable whether such a low cost could be improved or equalled by Tavener's process. However, it is worthy of investigation to see whether this process would not yield a higher percentage of saving of the metals.

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